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ACCIDENTS FROM FALLS OF ROCK OR ORE AT METAL AND NONMETALLIC MINES

Metal- and Nonmetallic-Mine Accident-Prevention
Course—Section 2
(Revised July 1955)

U. S. DEPARTMENT OF THE INTERIOR
March 3, 1849

U. S. Bureau of Mines
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Metal- and Nonmetallic-Mine Accident-Prevention Course—Section 2
(Revised July 1955)

U. S. DEPARTMENT OF THE INTERIOR
March 3, 1849

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Circular 92

UNITED STATES DEPARTMENT OF THE INTERIOR
Fred A. Seaton, Secretary
BUREAU OF MINES
Marling J. Ankeny, Director

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ACCIDENTS FROM FALLS OF ROCK OR ORE AT METAL AND NONMETALLIC MINES[1]

Metal- and Nonmetallic-Mine Accident-Prevention Course—Section 2

Revised by

Edward Thomas[2] and Roy G. Stott[3]

PURPOSE AND SCOPE

The first metal-mine accident-prevention course was prepared and published in 1942–45 as a series of seven miners' circulars (Nos. 51–57). The scope of the course has been broadened, revised, and brought up to date and is being published as a similar series of seven miners' circulars (Nos. 51–57, revised) on accident prevention in metal and nonmetallic mines. These circulars are:

Accident Statistics (Miners' Circular 51), dealing with general statistics on accidents and injuries at metal and nonmetallic mines, including causes, costs, and the use of investigations and reports of all accidents.

Falls of Rock or Ore (Miners' Circular 52), discussing the selection of mining methods to minimize the hazards of falling and sliding ground, the use of various types of support, and the protection of employees from falls of ground.

Hoisting and Haulage (Miners' Circular 53), presenting the hazards of hoisting and haulage at metal and nonmetallic mines and means of preventing accidents.

Explosives (Miners' Circular 54), giving information on accidents and injuries due to storing, handling, and using explosives in metal and nonmetallic mines and precautions for preventing them.

Fires, Gases, and Ventilation (Miners' Circular 55), explaining the causes of fires in metal and nonmetallic mines and the measures used to prevent, control, and extinguish them; describing gases found in mines and methods of detection and personal protection; and discussing the necessity for and standards of proper ventilation.

Electrical and Mechanical Hazards (Miners' Circular 56), covering accidents and injuries from electricity and machinery and their prevention and injuries from falls of persons.

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[1] Work on manuscript completed July 1955.
[2] Mining health and safety engineer, Bureau of Mines.
[3] Mining health and safety engineer, Bureau of Mines.

Health and Miscellaneous Hazards (Miners' Circular 57), including data on dust hazards; means of protection and sampling devices; protective clothing and equipment; and illumination, supervision, discipline, and safety training for employees in metal and nonmetallic mines.

These seven circulars do not contain all the material that may be desired on every phase of accident prevention at metal and nonmetallic mines, but they will serve as basis for discussion. To these may be added supplementary material of particular interest in the field where the course is utilized. This accident-prevention course, offered to the mining industry by the Bureau of Mines, is compiled from studies by the Bureau and experience and knowledge gained by its engineers, to which is added information on safe mining practices made available by mining companies and their officials.

This is the second in the revised series of circulars covering various phases of accident-prevention at metal and nonmetallic mines. This section deals with the occurrence of accidents and injuries caused by falls of rock or ore, their causes, and suggests preventive measures that may be employed to guard against and reduce the frequency and seriousness of injuries from this source.

ACKNOWLEDGMENTS

Much of the original text is retained in the revised course on Accident Prevention in Metal and Nonmetallic Mines.

The revision was under the general supervision of James Westfield, Assistant Director—Health and Safety, William J. Fene, chief, Division of Safety, and Simon H. Ash, former chief, Safety Branch, Bureau of Mines. This section was revised by Edward Thomas and Roy G. Stott; supplemental material was supplied by the following Bureau engineers:

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ACCIDENTS FROM FALLS OF ROCK OR ORE AT METAL AND NONMETALLIC MINES

INJURY STATISTICS

More fatal and nonfatal injuries at metal and nonmetallic mines (underground and opencut) result from falls and slides of rock or ore than from any other cause. Falls of rock or ore at underground mines (1932–54) accounted for 34.2 percent of the fatal and 16.1 percent of the nonfatal injuries. Falls or slides of rock or ore at opencut mines (1932–54) accounted for 26.0 percent of the fatal and 8.2 percent of the nonfatal injuries.

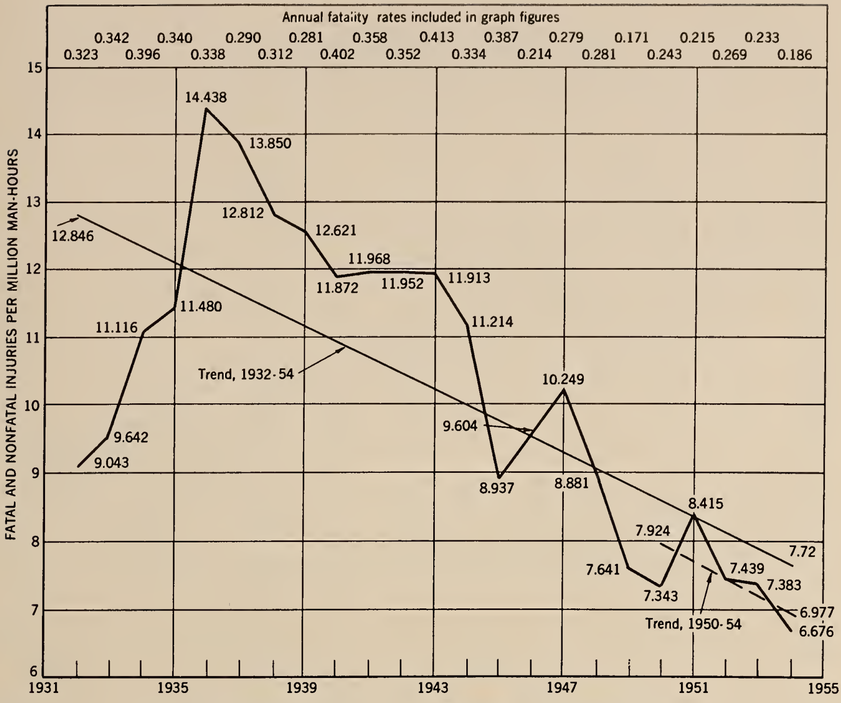

Figure 1.—Injury-Frequency Rates and Trends From Falls of Rock or Ore at Underground Metal and Nonmetallic Mines, 1932–54.

Accidents reported by mining companies to the Bureau of Mines show that injuries (fatal and nonfatal) from falls and slides of rock or ore at underground and opencut mines have declined from an average of 1,981 a year between 1932 and 1941 to 1,508 a year between 1942 and 1951 or 23.7 percent. In these decades injuries from all causes declined from an average of 12,039 a year to 9,780 or 18.8 percent. The total injuries from all causes are shown, and the injuries due to falls and slides of ground are enumerated both as to number and frequency rate in tables 1 and 2.

The injury-frequency trend from falls or slides of rock or ore at underground mines (fig. 1) was downward, with a decrease of 39.9 percent, during 1932–54. During 1950–54 the trend was also downward, with a decrease of 12.0 percent. For 1932–54 there were 1,043 fatal and 35,076 nonfatal injuries (total, 36,119), which is 16.4 percent of the total injuries from all causes. During 1950–54 there were 150 fatal and 4,711 nonfatal (total, 4,861) injuries, or 14.4 percent of the total. (See table 1.).

The injury-frequency trend from falls and slides of rock or ore at opencut mines (fig. 2) was downward, with a decrease of 83.6 percent, during 1932–54; during 1950–54 the trend was also downward, with a decrease of 47.3 percent. For 1932–54 there were 92 fatal and 1,542 nonfatal injuries (total, 1,634), which is 8.5 percent of the total injuries from all causes. For 1950–54 there were 14 fatal and 265 nonfatal injuries (total, 279) or 6.4 percent of the total injuries. (See table 2.)

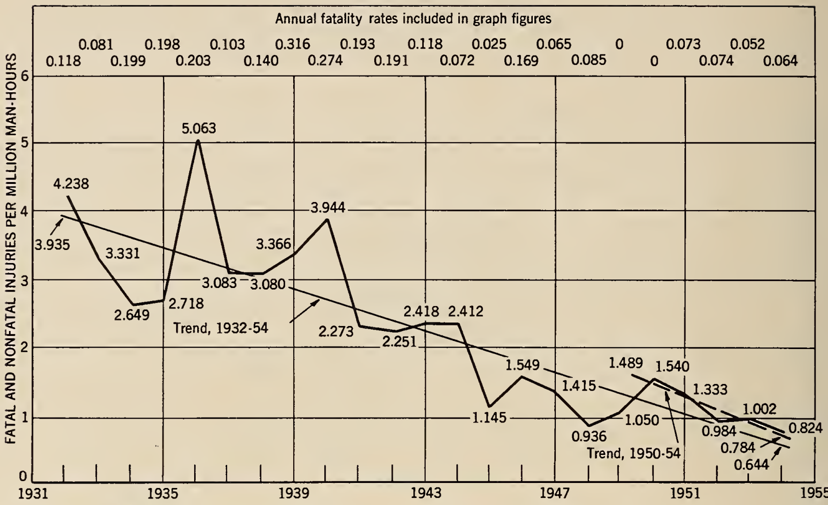

FIGURE 2.—Injury-Frequency Rates and Trends From Falls or Slides of Rock or Ore at Opencut Metal and Nonmetallic Mines, 1932–54.

TABLE 1.—*Injuries from falls of rock or ore at underground metal and nonmetallic mines, 1932–54*

[Prepared by the Branch of Accident Analysis, Division of Safety]

Year	Man-hours (millions)	Fatal				Nonfatal			
		All causes (total)	Falls of rock or ore	Percentage of total	Frequency rate [1]	All causes (total)	Falls of rock or ore	Percentage of total	Frequency rate [1]
1932	83.5	105	27	25.7	0.323	4,767	728	15.3	8.72
1933	81.8	90	28	31.1	.342	5,555	761	13.7	9.30
1934	101.1	101	40	39.6	.396	7,440	1,083	14.6	10.72
1935	141.1	154	48	31.2	.340	9,677	1,572	16.2	11.14
1936	177.7	185	60	32.4	.338	13,916	2,506	18.0	14.10
1937	210.3	207	61	29.5	.290	17,191	2,853	16.6	13.56
1938	166.8	145	52	35.9	.312	12,052	2,085	17.3	12.50
1939	181.6	157	51	32.5	.281	13,019	2,241	17.2	12.34
1940	201.6	200	81	40.5	.402	13,982	2,311	16.5	11.47
1941	207.0	201	74	36.8	.358	14,601	2,402	16.5	11.61
10-year:									
Total	1,552.5	1,545	522	--------	--------	112,200	18,542	--------	--------
Average	155.2	155	52	33.8	.336	11,220	1,854	16.5	11.94
1942	198.8	206	70	34.0	.352	12,682	2,307	18.2	11.60
1943	183.8	189	76	40.2	.413	11,758	2,114	18.0	11.50
1944	146.9	127	49	38.6	.334	9,100	1,599	17.6	10.88
1945	126.5	101	49	48.5	.387	7,213	1,081	15.0	8.55
1946	121.7	101	26	25.7	.214	7,817	1,143	14.6	9.39
1947	139.9	128	39	30.5	.279	8,632	1,395	16.2	9.97
1948	142.3	99	40	40.4	.281	7,849	1,224	15.6	8.60
1949	128.6	73	22	30.1	.171	7,242	960	13.3	7.47
1950	139.9	101	34	33.7	.243	6,995	993	14.2	7.10
1951	134.7	94	29	30.9	.215	7,212	1,104	15.3	8.20
10-year:									
Total	1,463.1	1,219	434	--------	--------	86,500	13,920	--------	--------
Average	146.3	122	43	35.6	.297	8,650	1,392	16.1	9.51
1952	133.9	115	36	31.3	.269	6,962	960	13.8	7.17
1953	128.9	98	30	30.6	.233	6,683	921	13.8	7.15
1954	112.9	74	21	28.4	.186	5,952	733	12.3	6.49

[1] Per million man-hours.

TABLE 2.—*Injuries from falls or slides of rock or ore at opencut metal and nonmetallic mines, 1932–54*

[Prepared by the Branch of Accident Analysis, Division of Safety]

Year	Man-hours (millions)	Fatal				Nonfatal			
		All causes (total)	Falls or slides	Percentage of total	Frequency rate [1]	All causes (total)	Falls or slides	Percentage of total	Frequency rate [1]
1932	8. 5	2	1	50. 0	0. 118	247	35	14. 2	4. 12
1933	12. 3	5	1	20. 0	. 081	370	40	10. 8	3. 25
1934	15. 1	15	3	20. 0	. 199	452	37	8. 2	2. 45
1935	20. 2	10	4	40. 0	. 198	529	51	9. 6	2. 52
1936	24. 7	14	5	35. 7	. 203	734	120	16. 3	4. 86
1937	29. 2	12	3	25. 0	. 103	864	87	10. 1	2. 98
1938	21. 4	11	3	27. 3	. 140	670	63	9. 4	2. 94
1939	25. 3	16	8	50. 0	. 316	691	77	11. 1	3. 05
1940	29. 2	23	8	34. 8	. 274	784	107	13. 6	3. 67
1941	46. 7	29	9	31. 0	. 193	1, 171	97	8. 3	2. 08
10-year:									
Total	232. 6	137	45	--------	--------	6, 512	714	--------	--------
Average	23. 3	14	4	32. 8	. 193	651	71	11. 0	3. 07
1942	52. 4	31	10	32. 3	. 191	1, 275	108	8. 5	2. 06
1943	50. 9	31	6	19. 4	. 118	1, 246	117	9. 4	2. 30
1944	41. 9	20	3	15. 0	. 072	1, 077	98	9. 1	2. 34
1945	39. 4	11	1	9. 1	. 025	854	44	5. 2	1. 12
1946	35. 6	15	6	40. 0	. 169	897	49	5. 5	1. 38
1947	45. 9	10	3	30. 0	. 065	969	62	6. 4	1. 35
1948	47. 0	20	4	20. 0	. 085	958	40	4. 2	. 85
1949	42. 7	6	--------	--------	--------	823	45	5. 5	1. 05
1950	36. 3	2	--------	--------	--------	854	56	6. 6	1. 54
1951	54. 8	18	4	22. 2	. 073	963	69	7. 2	1. 26
10-year:									
Total	446. 9	164	37	--------	--------	9, 916	688	--------	--------
Average	44. 7	16	4	22. 6	. 083	992	69	6. 9	1. 54
1952	53. 7	16	4	25. 0	. 074	893	49	5. 5	. 91
1953	58. 2	16	3	18. 8	. 052	914	55	6. 0	. 95
1954	47. 2	21	3	14. 3	. 064	658	36	5. 5	. 76

[1] Per million man-hours.

A review of injuries from falls or slides of rock or ore (including rolling and falling chunks) at the metal opencut mines of the Lake Superior district for 1950–54 [4] revealed that there were 2 fatal injuries, 25 nonfatal injuries, and 13,132 man-days lost or charged. This is 10.5 percent of all fatalities, 2.8 percent of the nonfatal injuries, and 7.0 percent of the total time lost or charged to all injuries during the period. (See table 3.)

A compilation of injuries from falls of rock or ore at the underground metal mines of the Lake Superior district for 1950–54 [5] (table 4) shows that there were 21 fatal injuries, 306 nonfatal injuries, and 144,536 man-days lost or charged. This is 33.3 percent of the total fatalities occurring at the underground mines, 11.0 percent of the lost-time nonfatal injuries, and 30.0 percent of the time charged to all injuries during the 5-year period.

Factors that influence the number of injuries and their frequency in respect to the amount of exposure of workers to the hazard of rock falls are the number employed, the time worked, and the relative

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[4] Lake Superior Mines Safety Council, Review of Accidents in the Lake Superior District: Proc. Annual Conf., 1950–54.

[5] See footnote 4.

TABLE 3.—*Injuries from falls or slides of rock or ore at opencut metal mines, Lake Superior district, 1950–54* [1]

Year	Man-hours (millions)	Fatal				Nonfatal				Fatal and nonfatal	
		All causes (total)	Falls of rock or ore	Percentage of total	Frequency rate [2]	All causes (total	Falls of rock or ore	Percentage of total	Frequency rate [2]	Man-days lost-time	Severity rate [3]
1950	19.24	5	1	20.0	0.052	177	9	5.1	0.47	6,175	0.321
1951	22.73	3	1	33.3	.044	198	8	4.0	.35	6,266	.276
1952	21.44	4				160	4	2.5	.19	230	.011
1953	24.72	3				223	2	.9	.08	358	.014
1954	17.81	2				131	2	1.5	.11	103	.006
Total	105.94	17	2			889	25			13,132	
5-year average	21.19	3.4	.4	10.5	.019	178	5	2.8	.23	2,626	.123

[1] Lake Superior Mines Safety Council, Proc. Annual Conf., 1950–54.
[2] Rate per million man-hours.
[3] Rate per thousand man-hours.

TABLE 4.—*Injuries from falls of rock or ore at underground metal mines, Lake Superior district, 1950–54* [1]

Year	Man-hours (millions)	Fatal				Nonfatal				Fatal and nonfatal	
		All causes (total)	Falls of rock or ore	Percentage of total	Frequency rate [2]	All causes (total)	Falls of rock or ore	Percentage of total	Frequency rate [2]	Man-days lost-time	Severity rate [3]
1950	20.78	9	7	77.8	0.337	548	54	9.9	2.60	44,895	2.160
1951	25.27	12	6	50.0	.237	627	64	10.2	2.53	42,023	1.663
1952	22.82	23	5	21.7	.219	561	57	10.2	2.50	32,977	1.445
1953	23.98	8	2	25.0	.083	569	59	10.4	2.46	15,004	.626
1954	23.48	8	1	12.5	.212	460	72	15.7	3.06	9,637	.410
Total	116.33	60	21			2,765	306			144,536	
5-year average	23.27	12	4	33.3	.171	553	61	11.0	2.62	28,907	1.261

[1] Lake Superior Mines Safety Council, Proc. Annual Conf., 1950–54.
[2] Rate per million man-hours.
[3] Rate per thousand man-hours.

numbers of experienced and inexperienced men working at any time, as well as several others, including the strata conditions, methods of mining, etc.

Improvements and modifications in mining methods have reduced the number of fatal and nonfatal injuries from rock falls in some mines since about 1937, and in recent years the reduction has been sufficient to affect the injury rate from falls of rock for all metal mines. The changes made at particular mines were necessary to prevent injury to the men employed, and in many instances they have had beneficial effects on production and costs aside from the saving in injuries and the expense connected therewith. Continued progress is to be expected as these safer methods are adopted in other mines or other effective modifications are devised. Where the method of mining is not suitable to the ore deposit, the men are exposed to excessive or unnecessary risks and will continue to be until the method is modified or changed.

The method of mining is decided upon by management, so theirs is the initial responsibility in taking into account the safety factors involved when determining the most economical way of extracting the ore. Mining is a competitive business, and margins of profit at times are narrow—so narrow in some instances that the cost of a single fatality may wipe out that margin for an appreciable period of time.

A certain stoping method may be highly desirable as to simplicity, low cost, and high production rate, but its use may be impossible or impractical due to limitations set by the physical characteristics of the ore deposit as they affect support. If the stopes cannot be kept open while the ore is being extracted, it is obvious that a method otherwise desirable cannot be employed.

Prevention of accidents from falls of ground is possible only if the method affords adequate support for the ground and protection to the employees in the working places. Accident prevention does not cease with the adoption of a safe mining method, as accidents will occur and injuries will result with any method unless safe practices are employed in every detail of the work. Nevertheless, if the general method is not the safest one to use in a particular instance, successful accident prevention is impossible, no matter how much attention is given to details of practice.

Metal-mining companies frequently acquire properties in which development work has been done in a small way and with a minimum regard for anything but to obtain ore or to determine the extent of the ore deposit. These steps seldom are retraced to bring the development workings up to normal standards; consequently, many metal mines start under a handicap. Until the extent of the ore deposit and grade of ore are determined, minimum expenditures are continued. Where deposits have been drilled or otherwise thoroughly explored, a safe plan for the development and extraction of the ore can usually be decided upon before actual opening of the mine is begun.

MINING METHODS AND SUPPORT OF GROUND

FACTORS AFFECTING CHOICE OF MINING METHOD

The importance of selecting the proper mining method for any particular ore body so as to minimize accident hazards has long been appreciated. Factors of practical operation and economic ore recovery must be considered with that of safety before a decision can be made on the method to be employed.

A number of considerations of major importance must be taken into account, including:

1. Characteristics of the ore body.
    - (*a*) Size of ore body.
    - (*b*) Shape of ore body.
    - (*c*) Character of wall rock and other geological features.
    - (*d*) Character of ore.
    - (*e*) Dip of the vein and of the containing strata.
    - (*f*) Depth of ore body.
    - (*g*) Proximity to other active or abandoned ore bodies.
2. Surface conditions and ownership.
3. Grade of ore to be mined.
4. Metallurgical treatment of ore.
5. Availability of labor.
6. Availability and handling of supplies.
7. Water hazards.
8. Fire hazards.
9. Accident hazards.
10. Transportation.
11. Mechanization possibilities.
12. Ventilation.

The method of mining is usually determined by considering the factors that promise the greatest economy in the cost of production. Personnel safety is an important factor in mining economy; its consideration is vital in the selection and pursuit of any mining method.

CLASSIFICATION OF STOPING METHODS

Stoping methods are classified according to the method of support as follows:

A. Stopes naturally supported.
    1. Open stopes without pillars.
        (*a*) Open stopes in small ore bodies.
        (*b*) Sublevel stoping.
    2. Open stopes with pillar supports.
        (*a*) Casual pillars.
        (*b*) Room (or stope) and pillar (regular arrangement).
B. Stopes artificially supported.
    3. Shrinkage stoping.
        (*a*) With pillars.
        (*b*) Without pillars.
    4. Cut-and-fill stoping.
    5. Stulled stopes in narrow veins.
    6. Square-set stoping.
C. Caved stopes.
    7. Caving (ore broken by induced caving).
        (*a*) Block caving, including caving to main levels and caving to chutes or branched raises.
        (*b*) Sublevel caving.

8. Top slicing (mining under a mat that, together with caving capping, follows the mining downward in successive stages).

D. Combinations of supported and caved stopes (as shrinkage stoping with pillar caving, cut-and-fill stoping with top slicing of pillars, etc.).

E. Modifications of stoping methods.

Basically, the stoping method or methods that can be applied to a given ore body depend on the requirements for support of the stope; the maximum area or span of the back and walls that will be or can be made to be self-supporting during the removal of the ore; the nature, size, and interval between supports required to maintain the backs and walls of the excavations; and the requirements for permanently supporting the overlying and surrounding country rocks and overburden to prevent or control their movement and subsidence. Variations of the principal methods of stoping may be based upon the direction or angle of working, sequence of operations, or methods of handling the broken ore.[6] The recent development of rock bolting as a means of ground support necessitates a reevaluation of some of the older mining methods and the feasibility of devising new methods in light of the new possibilities presented.

A. STOPES NATURALLY SUPPORTED

Stopes supported naturally are usually mined by overhand methods, although in occasional instances underhand methods (fig. 3) may be safer in narrow veins of weak ore and strong walls. In overhand mining the miners work near the back of the stope where they can examine the roof, take down loose ground, and are not exposed to danger of pieces falling from a height as in underhand mining.

1. OPEN STOPES WITHOUT PILLARS[7]

Naturally supported stopes are those in which no regular artificial method of support is used, although occasional cribs, props, rock bolts, or stulls may be used to hold local patches of insecure ground. (See fig. 4.) The simplest open stopes are those in which the entire ore body is mined out from wall to wall without leaving any pillars. Open stopes can be used only where the ore and wall rocks are firm.

In sublevel stoping the ore is excavated in open stopes, retreating from one end of the stope toward an entrance at the other end on each of a series of sublevels at intervals 25 to 50 feet apart vertically. The miners always work under undisturbed ground and have a reasonably safe avenue of retreat through subdrifts to the manway raises. Slabbing of the roof and walls at the worked-out end of the stope does not endanger the workmen and if the stope is mined back rapidly usually will not dilute the ore with waste. This system is applicable to wide, thick, irregular masses of ore that stand well without artificial support and have firm walls; also to thick tabular ore deposits that dip 50° or more. It is applicable in weaker ground that could be worked safely by other open-stoping methods, as the miners are always protected by solid ground overhead. Large open stopes are mined by this method where the ore is so soft it can be drilled with auger steel, and it can be applied where the ore is too

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[6] Jackson, Charles F., and Hedges, J. H., Metal-Mining Practice: Bureau of Mines Bull. 419, 1939, p. 223.

[7] Jackson, Charles F., and Gardner, E. D., Stoping Methods and Costs: Bureau of Mines Bull. 390, 1936, p. 5.

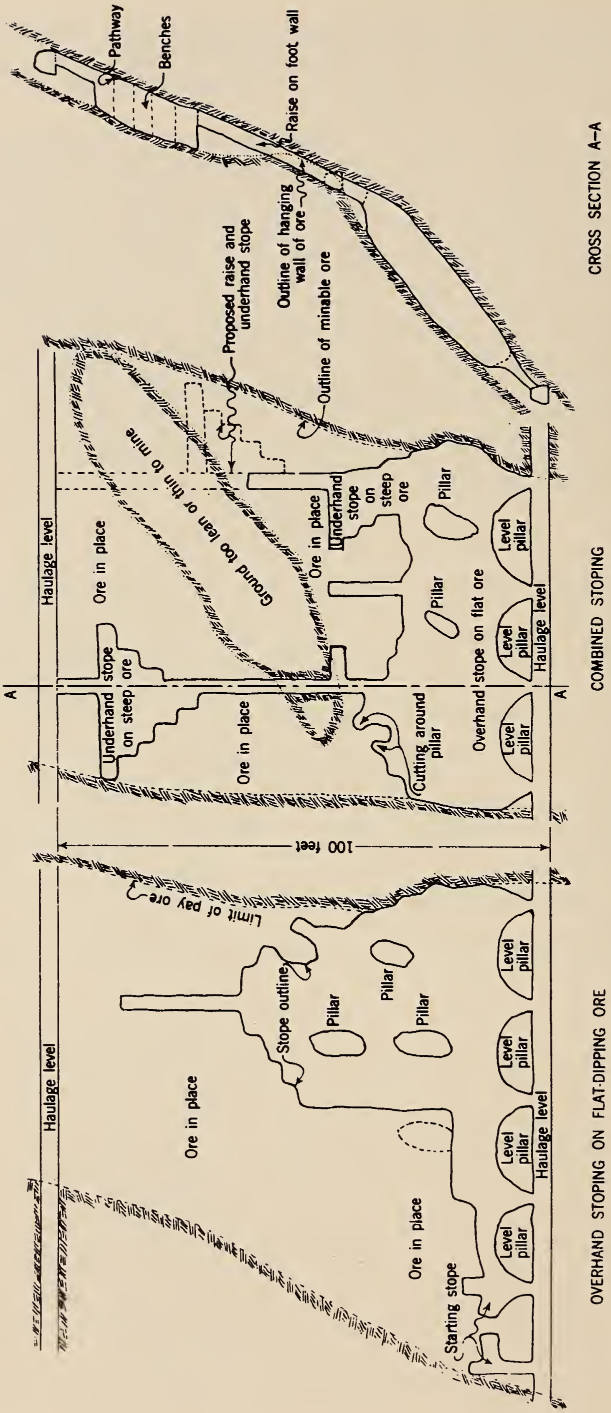

FIGURE 3.—Open Stopes Mined by Overhand and Underhand Methods.

weak to be mined safely by shrinkage stoping. In most instances rock bolts can be used to reinforce the skin area and the back and the walls of such openings without interference to mechanized handling of ore or to transportation of ore by gravity.[8] (See fig. 5.)

At a Canadian mine, in connection with a sublevel mining system in which long-hole drilling and blasting are done, the nearly vertical walls of weak rock are compacted before actual mining of the ore. The program is to drill extra holes along the wall (fig.6) and cut down the explosives concentration in all wall holes to minimize shattering. Tubular rock bolts are then cemented into the walls on 6-foot centers midway between the blasthole rings. At the sublevel elevation two 12-foot, slotted-type bolts are placed in each row. Between sublevels $1\frac{13}{32}$-inch down holes are drilled with a diamond drill, using a non-coring-type bit. In these holes 1-inch standard pipes 12 and 20 feet long are grouted into the wall as shown. A quick-setting aluminous cement is used.

An adaptation of sublevel stoping is sublevel long-hole benching; by this method irregularly shaped ore bodies that stand well can be safely and economically mined. As a precaution, miners use safety belts with ropes tied to pins set in holes drilled in solid rock.

2. OPEN STOPES WITH PILLAR SUPPORT[9]

In open stopes with pillar support, the length of unsupported span is reduced by leaving pillars of ore in place. Pillars may be of the casual type, their position and size being determined by ground conditions, or may be regular in size and arrangement to conform to a predetermined plan.

Open stoping is applicable to deposits of strong, firm ore having firm walls or ore and walls that are capable of reinforcement by rock bolts. In the narrower deposits (30 to 50 feet) the ore often can be stoped the full width of the deposit without leaving any pillars. In wider deposits it is usually necessary to leave pillars to reduce the length of unsupported span to prevent failure of the back. With casual pillars, sections of low-grade ore may be left advantageously. Casual pillars are used in the firmest type of ground and in deposits of variable thickness where their size and spacing can be proportioned to suit the conditions of the back and height of the ore. Within certain limitations the natural strength of the rock can be supplemented through proper placement of rock bolts.

The regular room- (or stope-) and-pillar system is particularly adaptable to thin, flat-lying beds where the roof must be supported definitely and permanently in places where it is not safe to rely on personal judgment for determining the amount of support required at individual local points.

Where extraction of pillars is contemplated, the pillars left in flat-lying beds in the development stage must be of sufficient area (fig. 7) to avoid being crushed by the weight of the cantilevered load of the totally mined pillars. The size of such pillars will depend upon the depth of cover to the surface and the ease with which the overlying rocks will break to the surface; otherwise, "squeezes" will

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[8] Thomas, Edward, Rock Bolting Finds Wide Application: Min. Eng., vol. 6, November 1954, pp. 1080–1085.

[9] Work cited in footnote 7, p. 9.

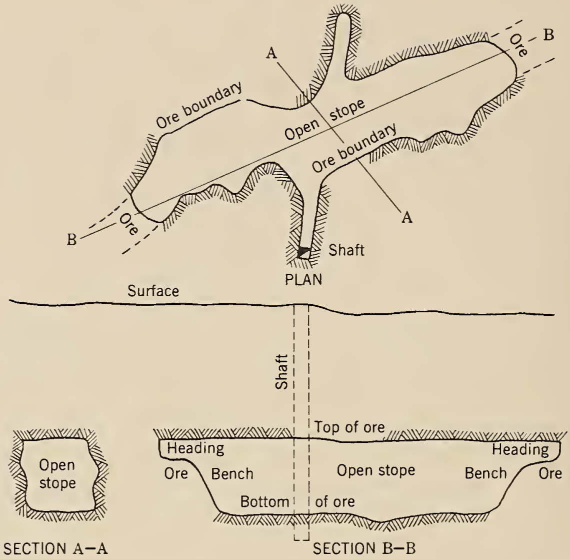

FIGURE 4.—Open Stoping Without Pillars in Small Ore Bodies.

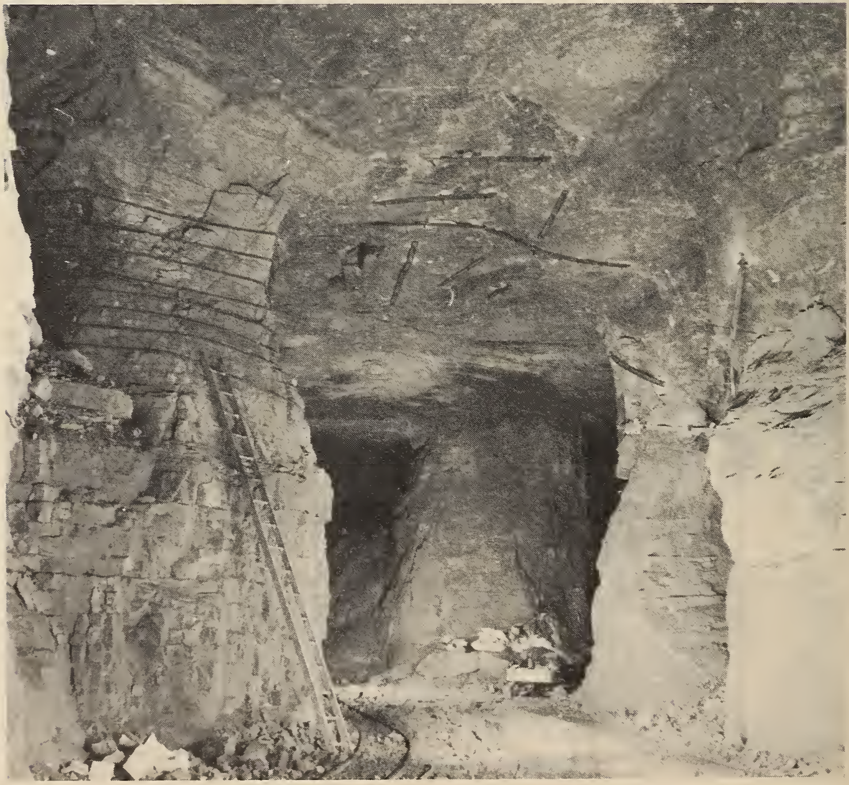

FIGURE 5.—Irregular Pillars in Open Stope; Back Supported With Rock Bolts and Channel-Bearing Plates; Sloughing Pillars Reinforced With Wire Rope and Rock Bolts.

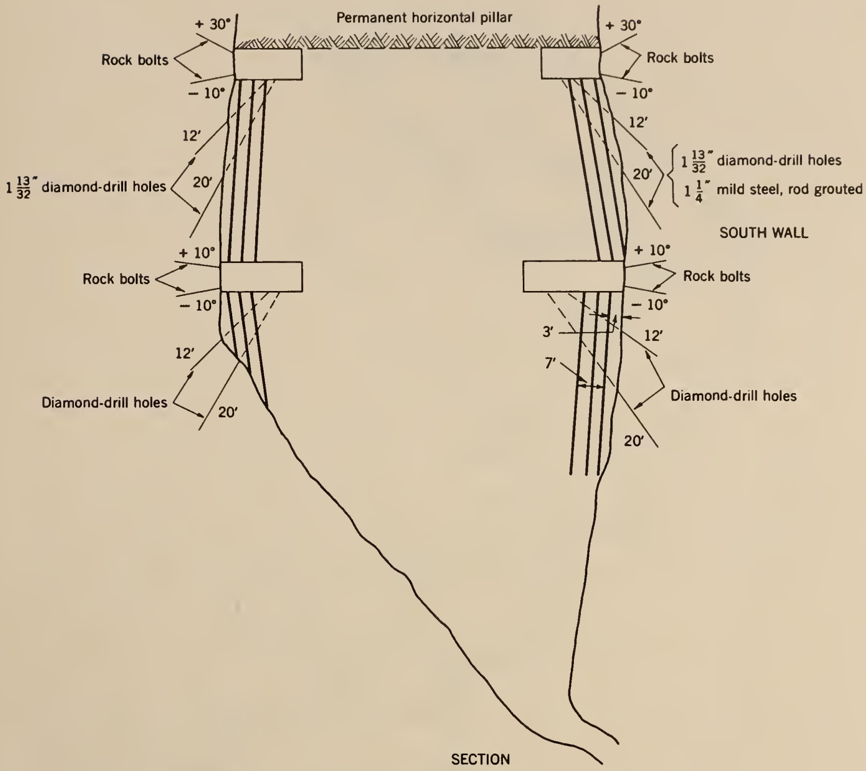

Figure 6.—Compacting Walls With Rock Bolts Before Mining Ore.

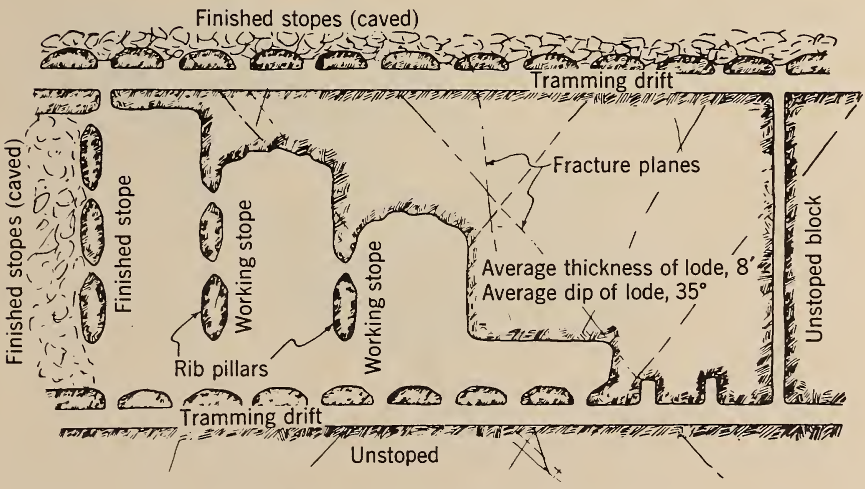

Figure 7.—Retreating System of Open Stoping With Rib Pillars. Section in Plane of Lode.

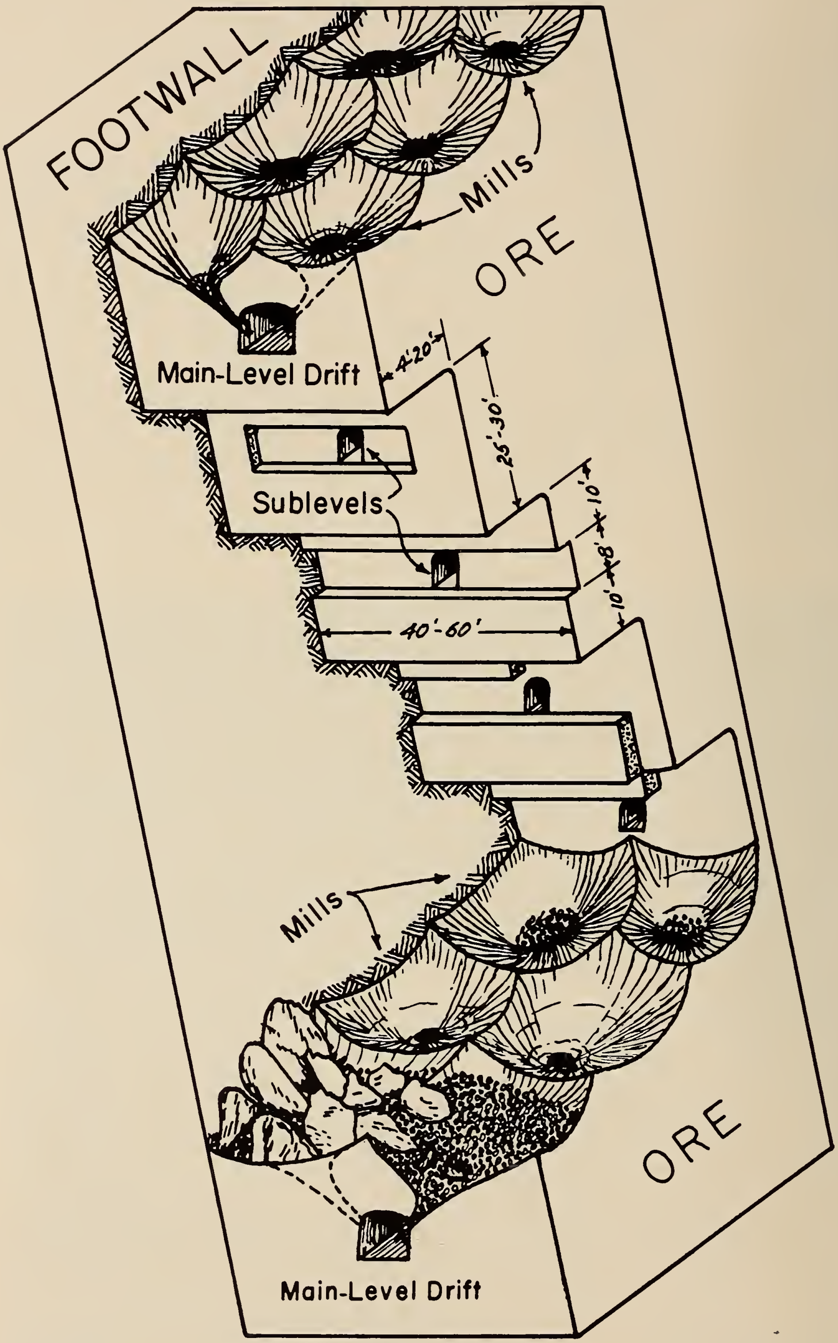

Figure 8.—Sublevel Stoping.

develop, ore will be lost, and a fractured roof will increase the hazards of falls of rock.

Where the ore lies in steeply pitching beds that are 20 to 60 feet wide, the rooms are developed by means of sublevel-stoping methods (fig. 8). Room-and-pillar sublevel stoping is chiefly applicable to narrow ore bodies but may be adapted to wider ore bodies by leaving larger pillars more closely spaced (fig. 9).

Advantages.—Where open-stope methods can be applied the cost per ton is low for the hard ore usually mined; mechanical loading and haulage methods can easily be applied; irregularities in the ore body can be followed; high total extraction can be secured by leaving lean ore in pillars and sorting out waste; ore can be removed as rapidly as it is broken; and development work is usually in ore. The back in work areas is usually accessible for inspection and scaling. Most drilling is done from the floor of sublevels, which provide adequate storage for supplies and equipment.

In some instances where pillars are developed by sublevel stoping, the pillars may later be extracted by caving or by long-hole blasting.

Disadvantages.—At times considerable ore must be left in pillars until robbing can be done later. It is often difficult to inspect and scale the back and walls. The accident rate is high except in the sublevel-stoping method. Eventual caving may make it difficult and dangerous to mine adjacent bodies of low-grade ore.

B. STOPES ARTIFICIALLY SUPPORTED

3. SHRINKAGE STOPING [10]

By this method the ore is mined in successive flat or inclined slices working upward from the level. After each slice is blasted down, enough broken ore is drawn off from below to provide a working space between the top of the pile of broken ore and the back of the stope (figs. 10 and 11). The remaining ore serves as a working floor from which to drill the succeeding cut and also provides considerable temporary support to the stope walls, justifying classification of this method as stoping artificially supported. Usually 40 percent of the broken ore has been drawn off when the stope has been mined to the top.

For shrinkage stoping the ore must be strong and firm, since the back under which the men are working is unsupported except for occasional props under patches of insecure ground. The walls must be firm so that they will not slough or collapse during or soon after broken ore is drawn off. In general, for shrinkage stoping the ore itself must be firmer and stronger than is required for sublevel stoping, while the walls may be somewhat weaker.

This method is applied most frequently to tabular deposits of moderate thickness, dipping at angles greater than 50°, in which few waste inclusions occur and which have fairly regular, strong walls.

Proper spacing of draw points and grizzly sublevels and the use of recently developed lighter drilling equipment, including that developed from air legs, have eliminated some of the objectionable features and hazards of shrinkage stoping.

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[10] Work cited in footnote 7, p. 9.

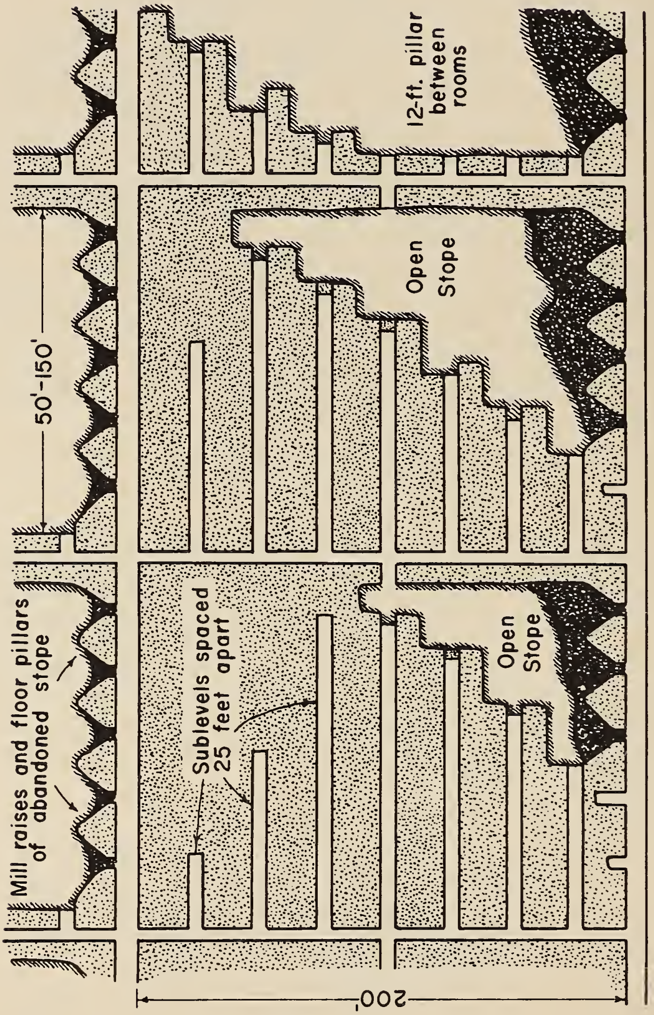
Mill raises and floor pillars of abandoned stope
50'-150'
12-ft. pillar between rooms
Open Stope
Sublevels spaced 25 feet apart
Open Stope
200'

Figure 9.—Room-and-Pillar Sublevel Stoping.

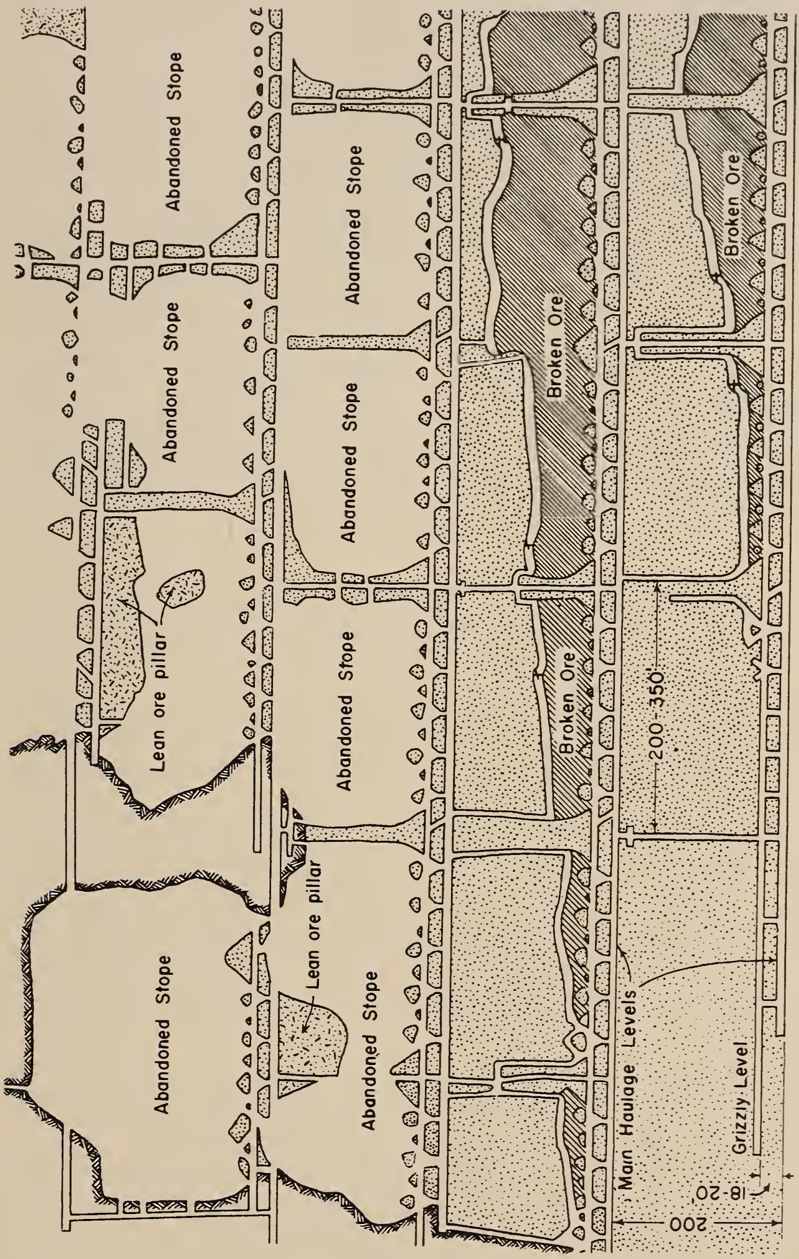

Figure 10.—Shrinkage-Stoping Method.

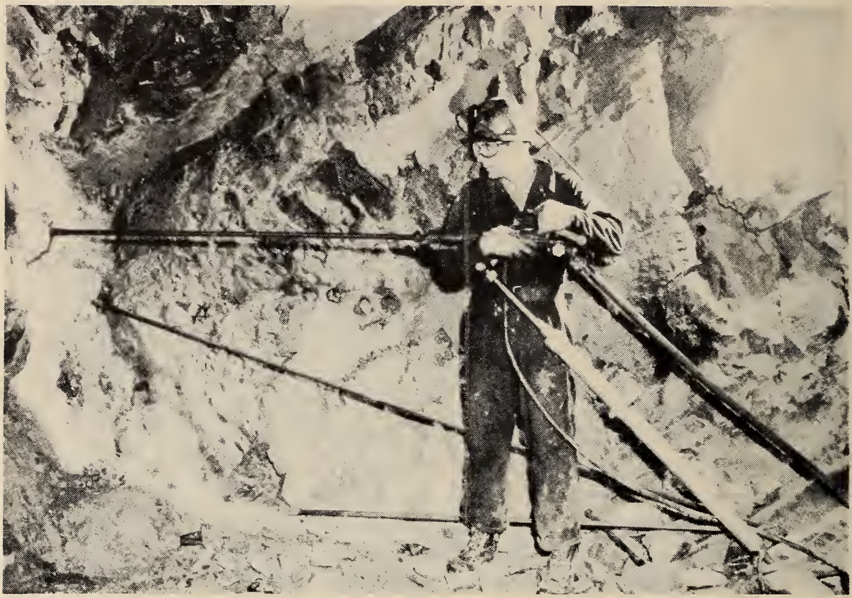

Figure 11.—Stoping With an Air Leg.

Shrinkage stoping is commonly used to outline the boundaries of ore blocks and to separate the ore from the wall rock preparatory to caving ore; to develop the initial openings of room-and-pillar open stopes; and to extract small ore-body offshoots of large ore bodies to be later mined by caving methods.

Advantages.—The shrinkage method requires only a small amount of preliminary development work and stope preparation. The broken ore supports the walls and eliminates the use of timber except as may be required to support occasional loose ground. The broken ore provides a floor upon which the men work close to the back so that scaling can be done efficiently and conveniently. Stopes can be worked rapidly, as little or no time is lost in handling and setting timber. In large stopes in good ground a number of crews can be worked in each stope, and supervision can be closely maintained. Long ore passes are not needed, and a reserve of broken ore can be held in the stopes that have been completed. Handling of ore is not required in the stopes, ventilation can be provided, and low costs are possible.

Disadvantages.—Inherent disadvantages are that the method has frequently been applied to deposits having walls that slough and cause excessive dilution of the ore; however, in many instances the wall-rock dilution may be minimized by the use of rock-bolt support to hold insecure wall rock. Approximately 60 percent of the ore is tied up until the stope is worked out. Chutes are usually spaced more closely than in some other methods, and manways left in the broken ore are more difficult to maintain. Where chutes are spaced widely, considerable shoveling or scraping may be required to clean out the stopes, and this may be dangerous as well as expensive. There is no opportunity to sort ore from waste in the stopes, and only those large chunks of ore that are on top of the ore pile can be bulldozed in the stope. Large chunks drawn into the chutes often present hazards as well as difficulties in handling them. If the broken ore hangs up and then suddenly collapses, miners may be drawn down into the ore. This difficulty is minimized to some extent when scrapers or slushers are used to transport the ore into the ore passes. Travel and handling of drilling equipment and supplies over uneven broken material are often difficult.

Where the dip and/or strike of a footwall is very irregular ore may be left during initial stoping. Later recovery of such tonnages has meant reentering stopes after the withdrawal of shrinkage ore. This practice has resulted in numerous fatalities and serious injuries caused either by rock falls from the hanging walls or by either the sliding or rolling of dislodged portions of the foot walls. Where such conditions prevail, a change in mining method is justified.

4. CUT-AND-FILL STOPING [11]

In cut-and-fill stoping, the ore is excavated by successive flat or inclined cuts or slices, working upward from the level, as in shrinkage stoping. However, after each slice is blasted down, all broken ore is removed, and the stope is filled with waste up to within a few feet of the back, thus providing permanent support to the walls and a new working floor (fig. 12).

Cut-and-fill stoping finds its best application in the mining of firm ore enclosed between walls, one or both of which are weak. The deposits may be tabular in shape or wide, thick ore bodies or masses. In general, it is suitable for mining deposits too irregular or with too weak walls for shrinkage stoping.

It is also used for extracting high-grade ore from veins too narrow to mine without breaking the walls to provide room. Where the ore and walls are broken separately, it is known as "stripping" or "resuing".

The filling material does not directly support the stope back, but only the walls; consequently, the ore itself must stand well enough, for a short time at least, to avoid the use of timber for auxiliary support. Farmin and Sparks[12] state that:

> To keep the cost of fill material at a minimum, to reduce the cost of handling fill material, to give greater flexibility for stoping irregular or perched ore bodies, to better support the stope walls and improve the safety of men, to improve dust and ventilation control, and to lower costs for chutes and manways, several mines have introduced sand-fill methods running the sand into stopes through pipelines.

Advantages.—Cut-and-fill stoping is especially applicable to the mining of either large or small ore deposits having moderately firm ore and weak walls. The filling prevents caving of weak walls and gives permanent support to the mined area. Irregular ore bodies can be mined, and offshoots of ore into the walls can be followed and mined. Inclusions of waste can be left in place or broken, sorted, and left in the stopes for filling. A high percentage of ore extraction

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[11] Work cited in footnote 7, p. 9.

[12] Farmin, Rollin, and Sparks, Carville E., Sand-Fill Method at Dayrock Resulted in 12 Benefits: Eng. and Min. Jour., vol. 152, September 1951, pp. 92–97.

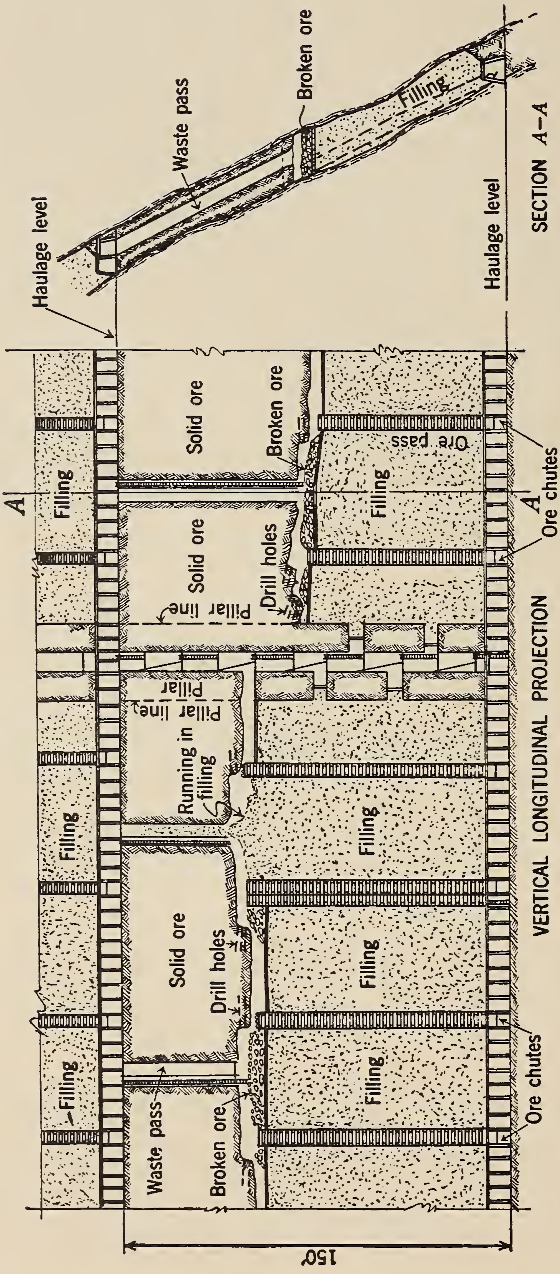

Figure 12.—Horizontal Cut-and-Fill Stoping.

is possible with little dilution of ore with waste. The ore is removed as it is broken so that the expense of unavailable ore is avoided. Secondary breaking, or bulldozing, can be done in the stope, and all material put in the chutes can thus be prepared for easy drawing and handling without extra risk. The stopes are easily ventilated, and the fire hazard is low, as little timber is required except in the stringer-set-and-fill variation of this method.

Disadvantages.—The method can be safely applied only where the ore itself is fairly strong, as the filling supports the walls alone and not the back. The rate of output is limited by delays caused by filling. A source of fill must be provided, and the breaking and handling of waste often are costly. The cost is higher than that for shrinkage stoping but lower than that for square-set stoping.

5. STULLED STOPES IN NARROW VEINS[13]

Stull timbering often is used in open stoping and in other stoping methods to support local patches of bad ground. However, a regular arrangement of stull timbers often is employed as a distinct method to support the hanging wall when stoping a relatively flat dipping tabular deposit and to support the hanging wall and stabilize the foot wall of a deposit that dips more than 45° (fig. 13). It is especially applicable where the immediate wall or walls are weak and will slab for several feet back to competent wall structure unless supported in some manner. If a slab from either wall is likely to fall when ore is blasted, diluting the broken ore, regular stulls are sometimes more economical than sorting wall rock from the ore in the stope. Where weak walls extend far back from the ore, timbering cannot be depended upon, but if the weak wall slabs are comparatively thin, regular stull timbering may act to steady the walls and be very effective. Filling may be introduced to provide better and more lasting support for heavy ground. The method is classified as cut-and-fill stoping if close filling is required and used.

Advantages.—The method gives support in mining thin, tabular deposits that have walls too weak for shrinkage stoping and where square-set timbering is impractical. The cost is lower than for square sets, and less labor is required to set the timbers.

Disadvantages.—A considerable amount of timber is required, and the stopes may have to be filled when finished to permit mining pillars or adjoining ore. Often adequate ventilation cannot be maintained economically. Frequent and careful inspections of the walls and stulls for security is necessary along travel routes and in work areas.

6. SQUARE-SET STOPING

Square-set stoping is that method of mining in which the walls and back of the excavation are supported by framed timbers in continuous lines in three directions at right angles to each other. The ore usually is mined in small, rectangular blocks just large enough to provide room for standing a set of timber. Ordinarily the stopes are mined out in floors or horizontal panels, and the sets of each successive floor are framed into sets of the preceding floor, but sometimes sets are mined out in a series of vertical or inclined panels. Square-set

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[13] Work cited in footnote 7, p. 9.

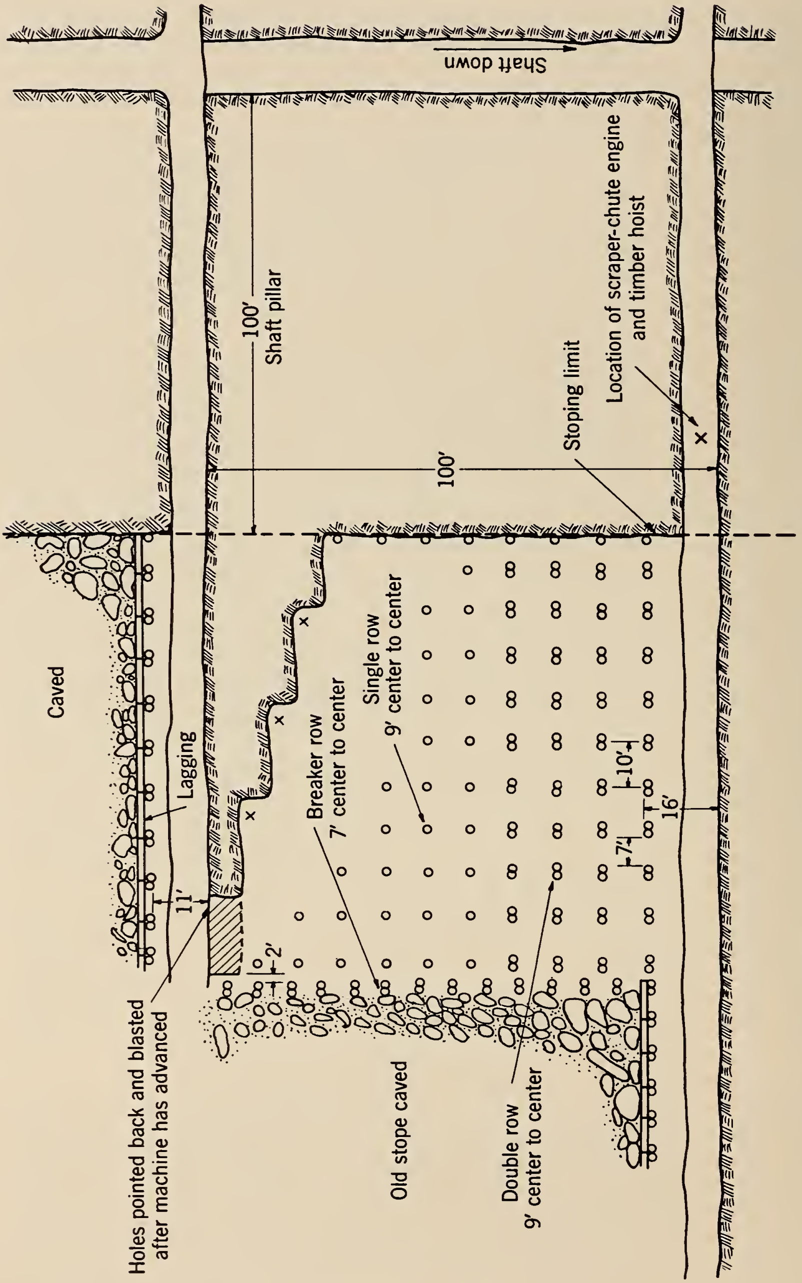
Shaft down
100'
Shaft pillar
100'
Stoping limit
Location of scraper-chute engine and timber hoist
Caved
Lagging
11'
Breaker row 7' center to center
Single row 9' center to center
2'
7'
10'
16'
Holes pointed back and blasted after machine has advanced
Old stope caved
Double row 9' center to center

FIGURE 13.—Stull Stoping, Conglomerate Lode, Michigan.

stoping usually is accompanied by filling of the stoped ground, and often in heavy ground the sets are filled with waste soon after they are put in, leaving only a small volume of unfilled stope at any time (figs. 14 and 15).

Square setting is primarily adapted to mining regular or irregular ore bodies where the ore or walls are too weak to stand for more than a brief time, even over short spans, and where caving and subsidence of overlying or adjacent ground must be prevented. It is particularly adaptable to the mining of very rich ore where complete recovery without contamination is important. From the standpoint of support alone, both temporary and permanent, the method is applicable to conditions under which no other satisfactory method has been devised. If the overlying strata may be allowed to cave and where the loss of some ore or some dilution with waste is not a serious objection, caved stope methods are adaptable to as bad ground as square setting or possibly worse ground. The cost of square setting and filling is

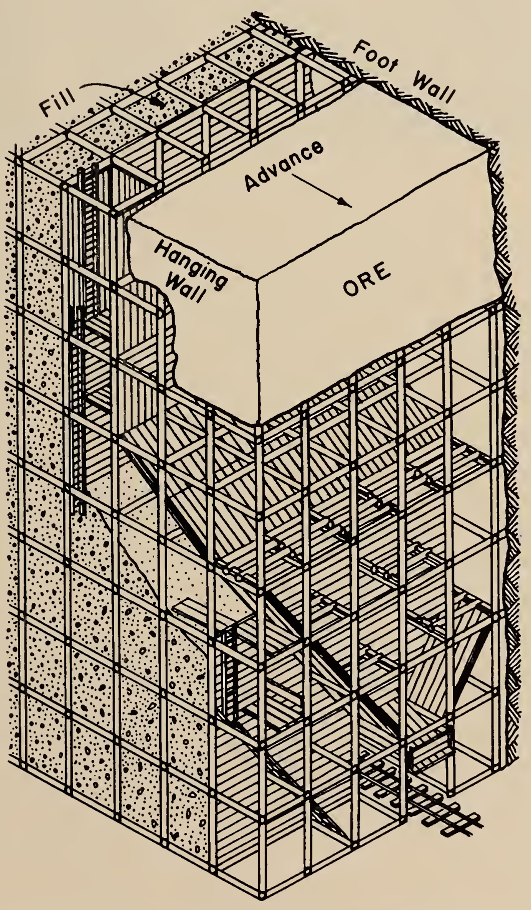

Figure 14.—Isometric Drawing of Square-Set Stope.

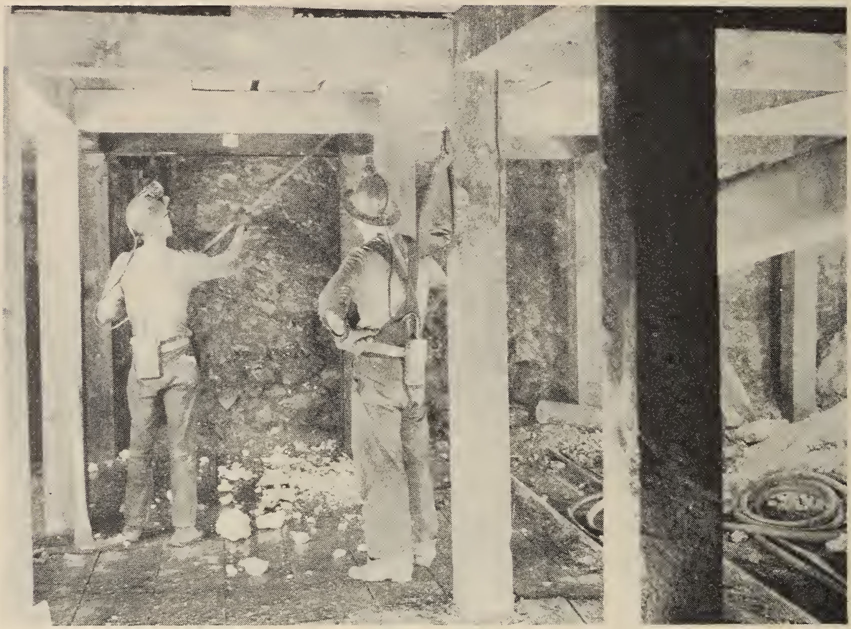

Figure 15.—Square-Set Stope.

warranted only in heavy ground requiring permanent support. The method is frequently used in combination with other methods, as for extracting pillars, for supporting sill floors of cut-and-fill or shrinkage stopes, and for slot-cutting for freeing ore blocks for block caving.

Advantages.—Weak ore with weak walls can be mined with little or no resultant caving and subsidence of the overlying and surrounding rocks. The method is flexible and can be employed on all dips in large or small ore bodies. Irregularities in the ore bodies can be followed and mined, and waste inclusions can be left in place or sorted out and used for filling in the stopes. Secondary breaking or bulldozing can be done in the stope, so that oversize material in the chutes is avoided. Stopes are ventilated with relative effectiveness; only a small area of ground need be open at any time; and the miners have relatively good overhead protection against falls of ground, as well as a floor on which to work.

Disadvantages.—The cost of mining, including framing, handling, and placing timber, is high, and the tonnage mined per man-shift is low; therefore, the method is not readily applicable to low-grade ore. Stoping progress is slow, resulting in a long life for individual stopes which may develop pressure, air-slacking, and attendant difficulties. Handling large quantities of timber and waste filling complicates underground transportation and increases the cost and hazards of haulage. Where large timbers are used accident hazards in handling and putting them in place are increased. The accident rate for square-set stoping is higher than that for other methods, due partly to the fact that it is employed in bad ground but also to the other risks such as those mentioned, as well as numerous others, including the difficulty in keeping floors in good condition, prevention of falling of slabs of rock or ore from the side walls, etc. The large amount of timber presents a greater fire hazard than most other stoping methods, and fires in the back-filled square-set stopes are very difficult to handle or control.

C. CAVED STOPES[14]

Caved stopes are of two distinct types. In the first, ore is broken by caving induced by undercutting a block of ore and isolating it from or weakening its connection to the surrounding ore or walls. Block caving and sublevel caving are methods of this type. In the second type the ore itself is removed by excavating a series of horizontal or inclined slices, while the overlying capping is allowed to cave and fill the space previously occupied by the ore. This is the top-slicing method.

7 (a). BLOCK CAVING[15]

In block caving, a thick block of ore is partly cut off from surrounding blocks by a series of drifts, one above the other, or by boundary shrinkage stopes; it is then undercut by removing a slice of ore or a series of slices separated by small pillars beneath the block. The isolated, unsupported block of ore breaks and caves under its own weight. The broken ore is drawn off from below, and as the caved mass moves downward it is broken further by pressure and attrition. The overlying capping caves and follows the broken ore downward. In the earliest applications of caving, the block was undercut on or immediately above the haulage level, and the caved ore was shoveled into cars in drifts driven or spiled through the cave, entailing heavy timbering and hand shoveling. In present practice the ore is caved to a number of branch raises from main-level haulage drifts well below the influence of the pressure induced by the caving block, and hand shoveling is virtually eliminated.

Natural forces are utilized to the highest degree for breaking the ore, transferring it to the haulage levels, and filling the mined-out areas. Lean ore and horses of barren waste are broken and drawn with the ore. The method is applied to large masses of ore that will cave when unsupported and will break finely enough to pass through the extraction raises as the caved mass is drawn downward. Bulldozing is usually an essential feature, and important advances have been made in the solution of accident hazards in this operation in recent years (fig. 16).

Several large mining companies have changed over from the chute and grizzly system to a slushing system of handling the caved ore

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[14] Work cited in footnote 7, p. 9.
[15] Work cited in footnote 7, p. 9.

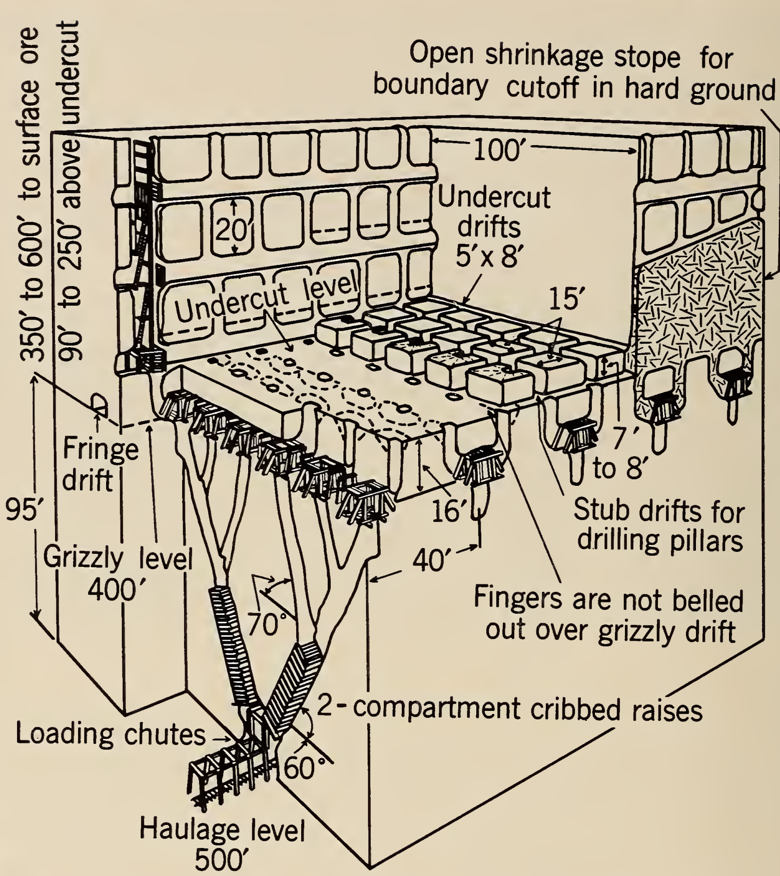
350′ to 600′ to surface ore
90′ to 250′ above undercut
Open shrinkage stope for boundary cutoff in hard ground
100′
20′
Undercut drifts 5′x 8′
Undercut level
15′
7′ to 8′
Fringe drift
95′
16′
Stub drifts for drilling pillars
Grizzly level 400′
40′
70°
Fingers are not belled out over grizzly drift
2-compartment cribbed raises
Loading chutes
60°
Haulage level 500′

Figure 16.—Perspective of Undercut Block-Caving Method.

(figs. 17 and 18). One company[16] reports several notable advantages of the slusher system as listed below:

1. Lower costs per ton of ore produced.
2. Higher efficiency—more tons per man shift.
3. Greater labor utilization and less travel time for men and bosses.
4. Closer supervision.
5. Ability to handle larger sizes of chunks and thereby reducing secondary blasting.
6. Fewer spills at loading points.
7. Better ventilation and dust control.
8. A marked reduction of the severity of accidents due to:
    - (*a*) Better footing.
    - (*b*) Less secondary blasting.
    - (*c*) No men working overhead of others.
    - (*d*) Reduced handling of supplies.
    - (*e*) Better cleanup.
    - (*f*) Improved communications.
9. Increased ore made available for mining from an established level.
10. Fewer openings to maintain.

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[16] Henderson, Robert, A Comparison Between Chute and Grizzly System and the Slusher System at the Climax Mine: AIME Tech. Pub. 1715, Min. Technol., vol. 8, No. 3, May 1944, 14 pp.

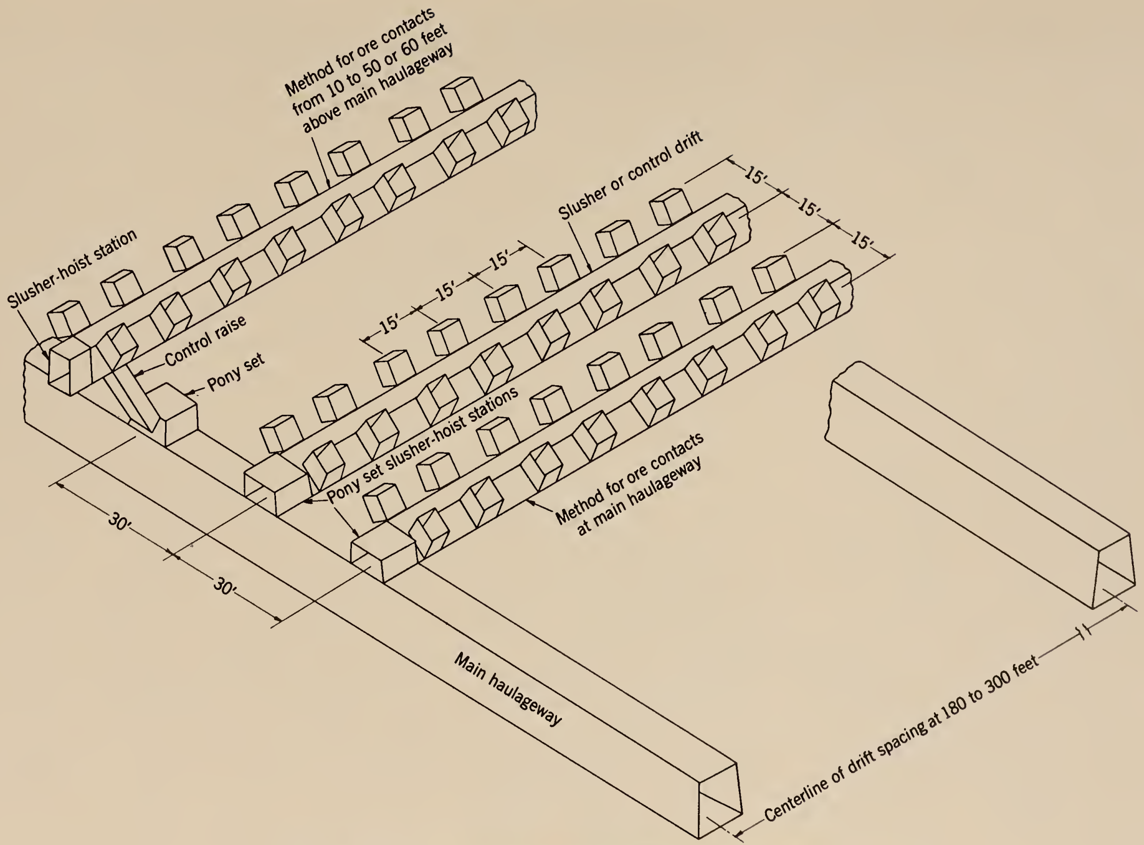

FIGURE 17.—Slushing Method to Main Haulage Drift.

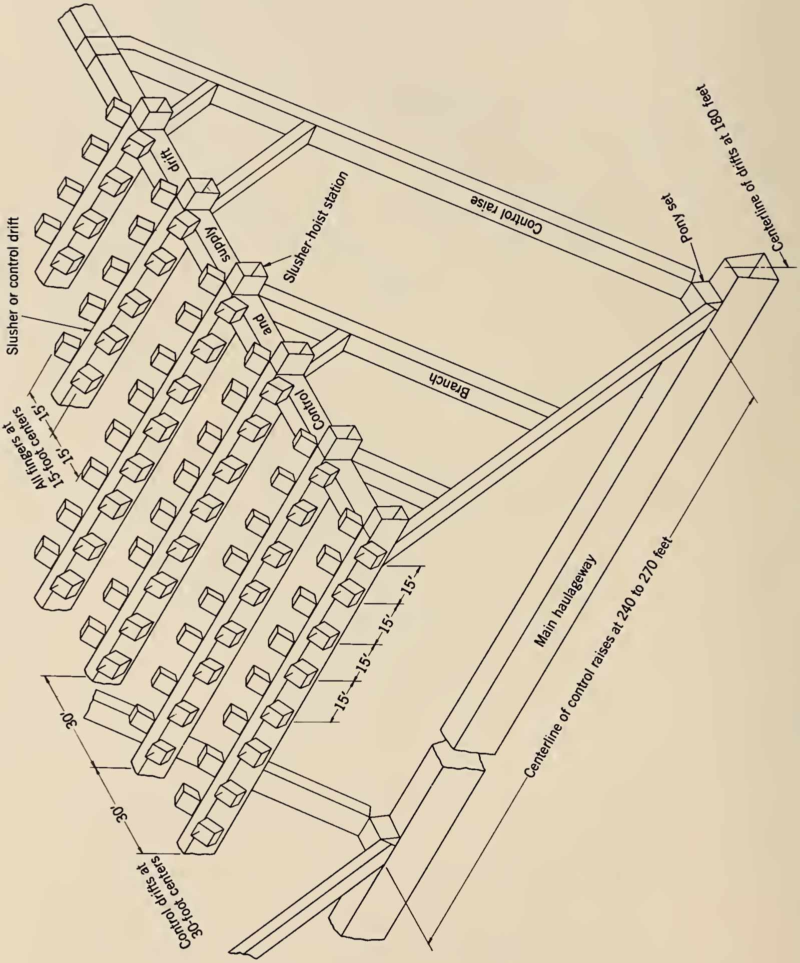
Slusher or control drift
All fingers at 15-foot centers
15'
15'
drift
supply
and
Control
Branch
Control raise
Slusher-hoist station
Pony set
Centerline of drifts at 180 feet
Main haulageway
Centerline of control raises at 240 to 270 feet
15'
15'
15'
15'
15'
30'
30'
Control drifts at 30-foot centers

FIGURE 18.—Slushing Method for High-Lying Ores.

Selection of caving method should be only after a study of all governing conditions, such as size of ore body, attitude, treatment demands, possible rock pressures, and cost and profit comparison of methods. The manner in which blocks of ore are caved and the sequence of retreat determines which method of block caving is used.[17] The present trend in shaping drawpoints and undercutting blocks for caving is by means of longhole drilling where miners work under a protected back.

Advantages.—The cost per ton is low, and a high rate of output is possible once the mine has been developed to the extraction stage. Various operations can be standardized to a high degree, with resulting high efficiency and safety. Well planned and operated block-caving systems have achieved low accident rates.

Disadvantages.—The expense and time needed for development before production are considerable. The method is nonselective, and marginal ore often is lost owing to waste dilution. Accident hazards at bulldozing chambers, grizzlies, and finger raises are severe; the dust hazard is likely to be difficult to control, and if the ore is highly siliceous the health hazard is likely to be extreme.

7 (b). SUBLEVEL CAVING[18]

In sublevel caving, relatively thin blocks of ore are caused to cave by successively undermining small panels. The ore body is developed by a series of sublevels spaced at vertical intervals of 20 to 60 feet. Timbered slice drifts are driven right and left opposite the raises to the limit of the block of ore to be mined. Caving back from certain drifts is begun and continued while other slices are being driven. Caving is accomplished by blasting out cuts from the drift face and retreating toward the raises. The broken ore is dragged by power scrapers into the raises. The bottoms of slices are covered with poles, lagging, or wire netting to form a light mat to hold back the gob when the next sublevel below is caved. This method is intermediate between block caving and top slicing.

In sublevel caving the ore is stoped partly by excavation and partly by caving. The method may be applied to smaller deposits than block caving but is also applicable to large deposits and to soft ores that will stand fairly well for a brief time over short spans with light timber supports but will cave when an appreciable area is undercut. The capping, and later the gob, should hang up over short spans long enough to permit removal of the caved ore beneath in safety and without undue dilution of the ore with waste. It can be applied where top slicing would be dangerous due to hanging up of the gob. Because of the relatively thin back and small units caved at one time, packing can be avoided when this method is used, and it is therefore applicable to softer, somewhat more sticky ores than is block caving. However, it is also applied to hard ores.

Advantages.—Sublevel caving can be used to mine smaller and more irregular deposits than with block caving. It can be used where block caving would be impractical and top slicing would be dangerous. Large inclusions of waste can often be left and caved with the gob to reduce dilution of the ore. Rapid, fairly complete extraction of the ore can be obtained at a low cost for mining heavy ground, and less timber is used than in top slicing. The accident rate can be held relatively low.

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[17] McNicholas, F. S., Classification of Block Caving and Draw Methods: Min. Eng., vol. 3, January 1951, pp. 33–38.

[18] Work cited in footnote 7, p. 9.

Disadvantages.—Some dilution occurs, and little sorting can be done; moreover, the recovery is usually not as complete as in top slicing. Considerable preliminary development is required, and ventilation must be provided under difficulties. There is a fire hazard from timber in the gob, and oxidation of timber gob may cause the air to attain disagreeably high temperatures. Rushes or slides of fine or muddy ore and gob material are often encountered, which are a serious accident hazard and cause delays in production.

Modifications made in a sublevel-caving system in a large iron mine are described in the published proceedings of the Lake Superior Mining Section of the National Safety Council for 1942.[19] By chang-

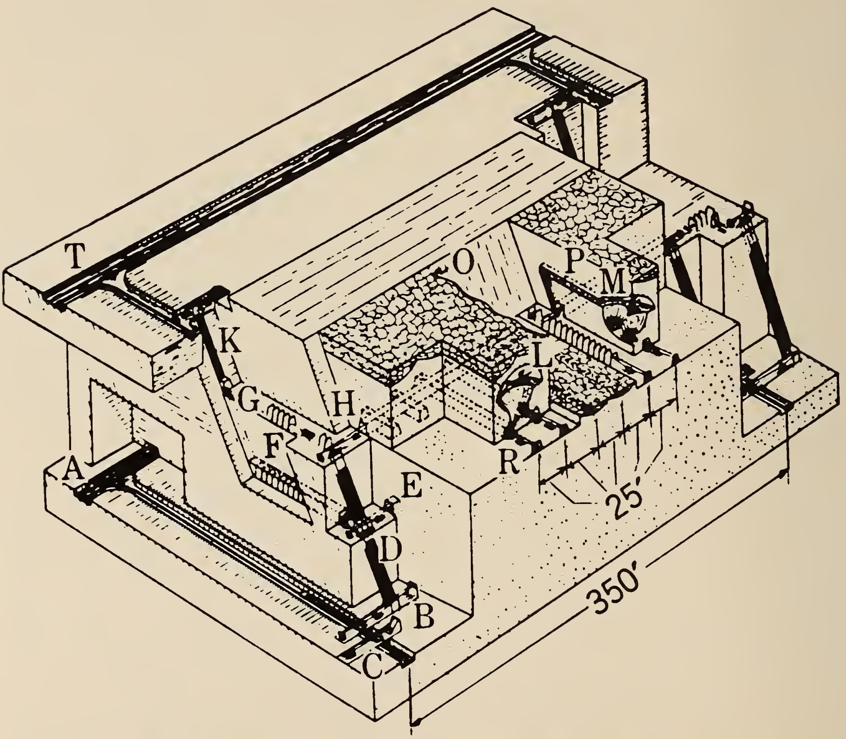

Figure 19.—Block Diagram, Showing Sublevel Caving and Tram-Car Loading, Using Slushers: *A*, Main haulage level; *B*, tram car loading drifts from west raise; *C*, tram car loading drifts from east raise; *D*, ore raise and manway; *E*, transfer drift—50-foot sub-starting development; *F*, ventilation and supply crosscuts, 50-foot sub; *G*, ventilation and supply crosscuts, 1,000-foot sub; *H*, transfer drift, 100-foot sub; *K*, ventilation and supply raise; *L*, cross section of stope; *M*, longitudinal section of stope and manway; *O*, caved gob; *P*, manway to stope; *R*, stoping crosscuts 20 to 120 feet long; *T*, former haulage level, now used for ventilation, manway, and supplies.

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[19] Fritz, D. E., The Elimination of Chutes for Tram-Car Loading at the Montreal Mine: Proc. Nineteenth Mine Safety Conf., Lake Superior Mining Sec., Nat. Safety Council, 1942, pp. 32–47.

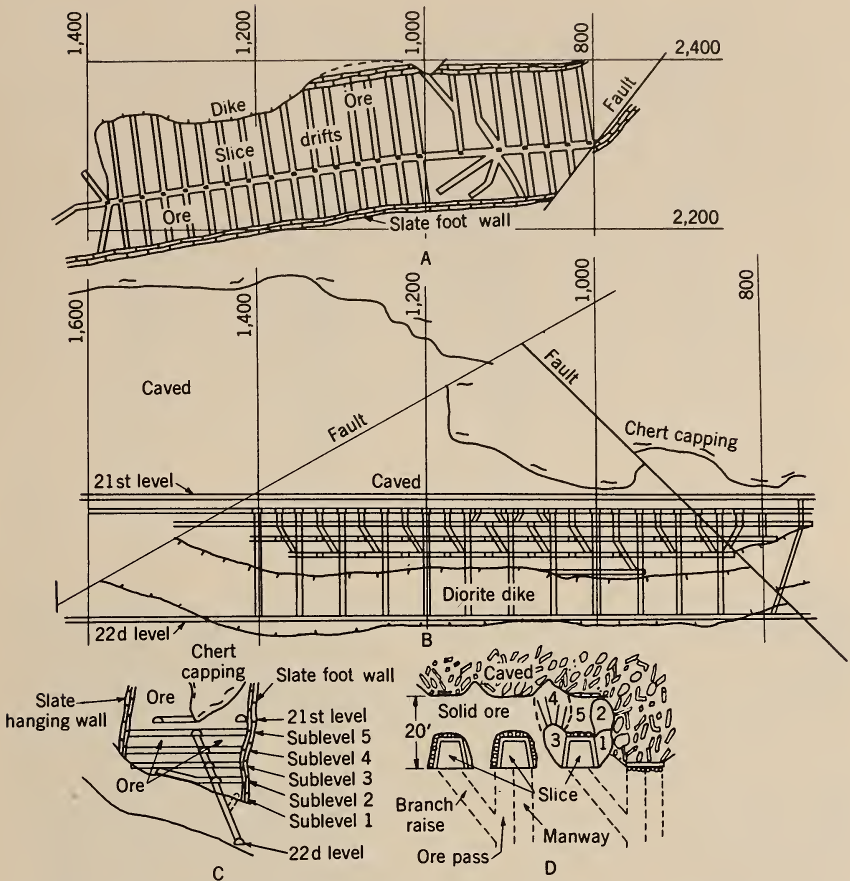

FIGURE 20.—Sublevel Caving: *A*, *B*, and *C*, Development of ore body for caving; *D*, cross section showing details of caving back, cuts blasted in order shown.

ing from grizzlies and chutes to slusher drifts numerous former hazards were eliminated. These hazards in the former method were encountered in the handling of ore in chutes, on the grizzlies, and in the ore raises. Haulage hazards were also greatly reduced. The modified system is illustrated in figure 19.

Figures 20 and 21 illustrate sublevel caving methods in iron mines.

8. TOP SLICING

In top slicing the ore is removed by excavating a series of timbered slices, one alongside the other, beginning at the top of the ore body and working progressively downward (fig. 22). The slices are caved by blasting out the supporting timbers, bringing the capping or overburden down on the bottom of slices that have previously been covered with a floor or mat of timber or wire fencing (fig. 23).

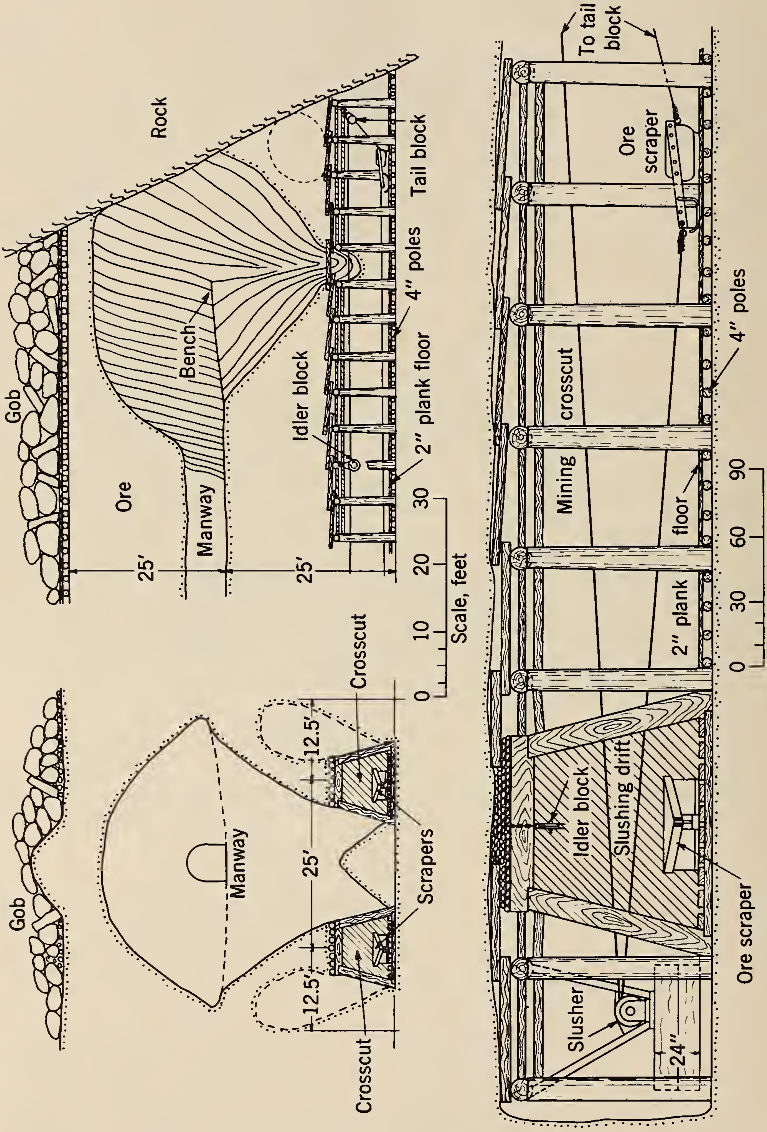

FIGURE 21.—Sublevel Caving, Large-Pillar Method.

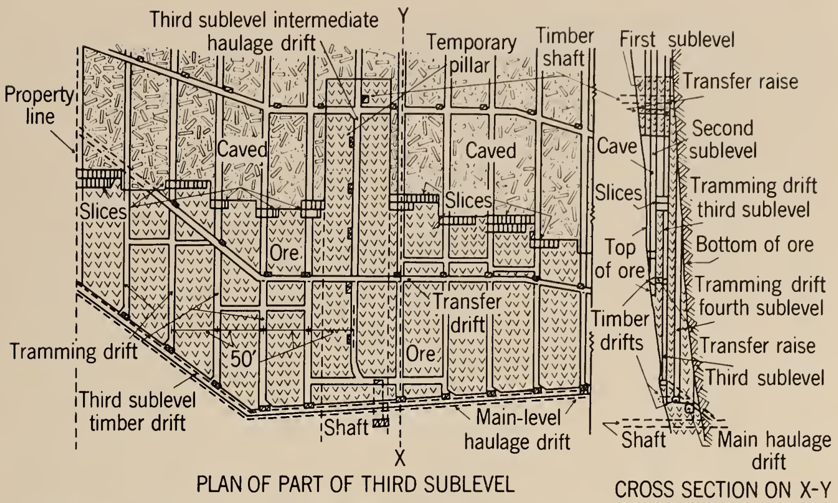

Figure 22.—Parallel Top Slicing, Scraping to Cars; Flat-Dipping Ore Body, 2 to 4 Sublevels Thick.

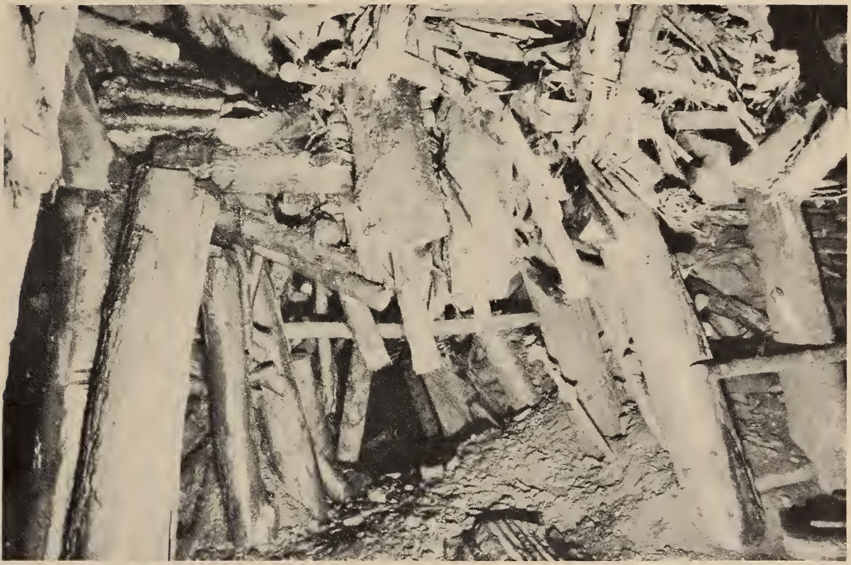

Figure 23.—Timber Mat in Top-Slice Stope.

The method is applicable to mining soft, weak ore that will stand unsupported for only a short time, even over spans of a few feet, and that is overlain by a capping that will break and cave as soon as the support is removed, tightly filling the space formerly occupied by the ore and leaving no holes. A capping that breaks into large blocks that arch over and hang up so as to leave open cavities in the gob would constitute a serious hazard in top slicing. The method is similar to sublevel caving, except that in top slicing the slice is mined right up to the floor of the caved slice above. In sublevel caving the slice drifts are separated by pillars that are mined on the retreat, whereas in top slicing the cuts are driven immediately alongside each other, and all ore is excavated.

Advantages.—Top slicing is a relatively safe method of mining heavy ground where the capping will break and follow as the ore is removed. It is economical where timber is plentiful and reasonable in cost. Irregular bodies of ore can be mined without loss of ore or dilution from the capping or the walls. Stopes may be closed and reopened with little or no loss of ground or heavy expense. Virtually all the ore is recovered if top slicing is conducted efficiently and a good mat is maintained. Excellent safety records may be attained with this method, provided care is taken in maintaining the timbering.

Disadvantages.—The method requires an adequate supply of timber and entails higher costs than block caving or shrinkage stoping. Except in deep mining of thin ore bodies, the surface cannot be maintained. Ventilation is difficult and requires special provision for each slice; high temperatures and humidities are likely to occur due to oxidation of the timber mat, and unless a reasonable amount of circulating air is maintained there may be oxygen-deficient air. Timbers in the mat, in the slices, and the supplies in the mine for current needs provide a fire hazard. Numerous working places and consequent development openings are required, often scattered over a considerable area. Much labor and time are required in handling and placing timber, lagging, and wire fencing. Timber handling over large working areas increases accident potentialities. If the capping breaks in large blocks that wedge together so as to leave a large, open space below them, there is a dangerous possibility of sudden collapse onto an active slice below.

D. COMBINATIONS OF SUPPORTED AND CAVED STOPES

The use of two or more stoping methods independent of each other does not constitute a combination method. However, when a stoping system is laid out in advance of operations wherein two different methods are employed as essential complements, one to the other, a combination method does exist. For example, where a series of alternating stopes and pillars is laid out, the stopes can be mined by shrinkage and then filled or mined by the cut-and-fill method, and the pillars can be mined by top slicing between the filled stopes. Each method, one a supported-stope method and the other a caved-stope method, is essential to complete extraction of the block of ore (fig. 24).

E. MODIFICATION OF STOPING METHODS

Recent mining practices, through the development of long-hole drilling, have been toward the mining of open stopes and the undercutting of caved stopes with miners working under a safe cover (fig. 25). Open blasthole stopes may be mined in their entirety without need of men entering the stopes. Radial blasthole end slicing has successfully replaced the open-stope sublevel bench or "Aldermac"

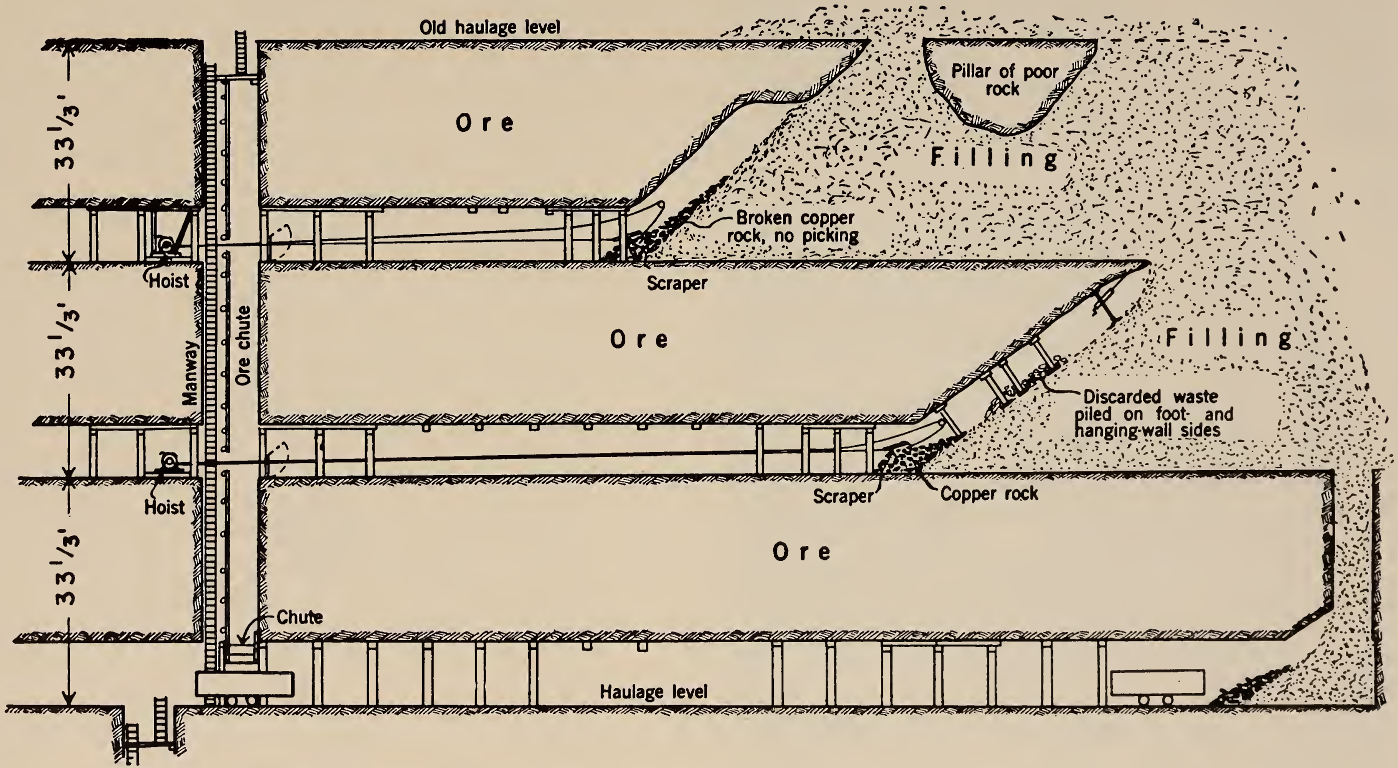

FIGURE 24.—Combination Method of Stoping; Sublevel, Inclined Cut-and-Fill Stoping.

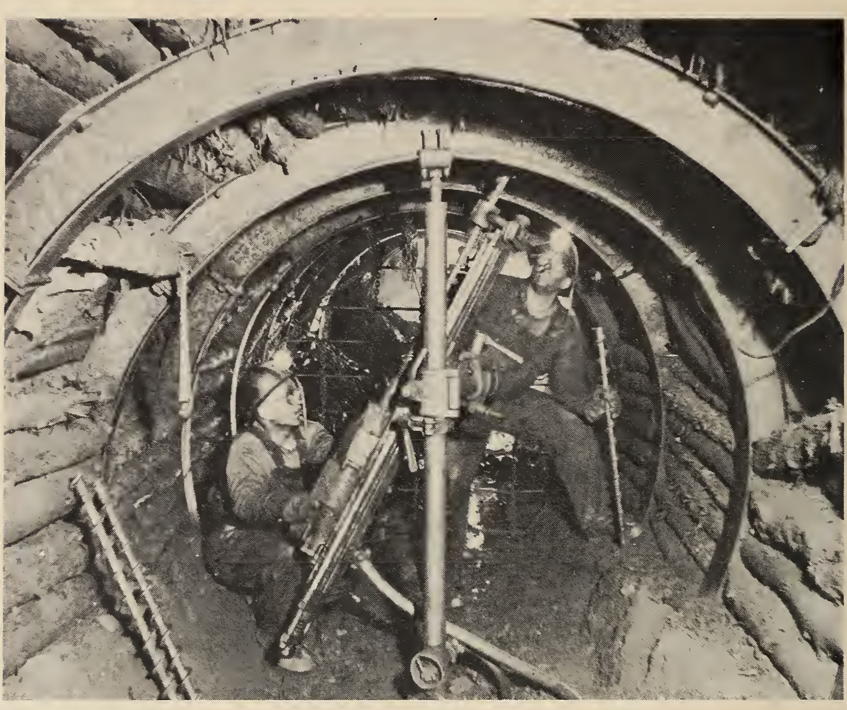

FIGURE 25.—Long-Hole Drilling From Cover of Slushing Sublevel to Induce Block Caving.
(Courtesy, The Cleveland-Cliffs Iron Co.)

method (fig. 26), and the "Horadiam" method of horizontal blastholes has been successfully introduced in shrinkage stoping.[20] In the latter method, horizontal radial blastholes are usually drilled from drilling raise cutouts. At one mine the drilling is from drilling raises (fig. 27) with the radial rings drilled 10 feet (vertically) apart. The horizontal blasthole method has been used to start induced block caving.

COMPARISON OF STOPING METHODS

Accident records of metal and nonmetal mines using different mining methods are compiled by the Branch of Accident Analysis, Bureau of Mines, from reports made by mines throughout the United States. These figures for the period 1943–52 have been combined in tables 5 and 6 to show the accidents from all causes associated with each type of stoping or mining method and those from falls of ground only.

Conclusions regarding relative safety of the different stoping methods must be drawn with care, because the accident frequency at certain mines may be affected by other factors than the stoping method itself. Any comparison between hazards of the different methods must consider the fitness of the method in the place where it is used. The facts are that mines using shrinkage stoping have had the highest rate of fatal and nonfatal accidents and that while timbered stoping gives protection in heavy ground, caving methods show lower rates of fatal and nonfatal accidents.

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[20] Douglas, R. S., Brehaut, C. H., Taylor, W. N., and Shannon, H. A., The New Horadiam Method of Mining at Copper Mountain: Trans. AIME, Min. Practice, vol. 163, 1945, pp. 42–72; AIME Tech. Pub. 1914, Min. Technol., vol. 9, No. 5, September 1945, 31 pp.

A comparison of accidents from three methods of caving have been combined in tables 7 and 8 from the accident records of mines in the

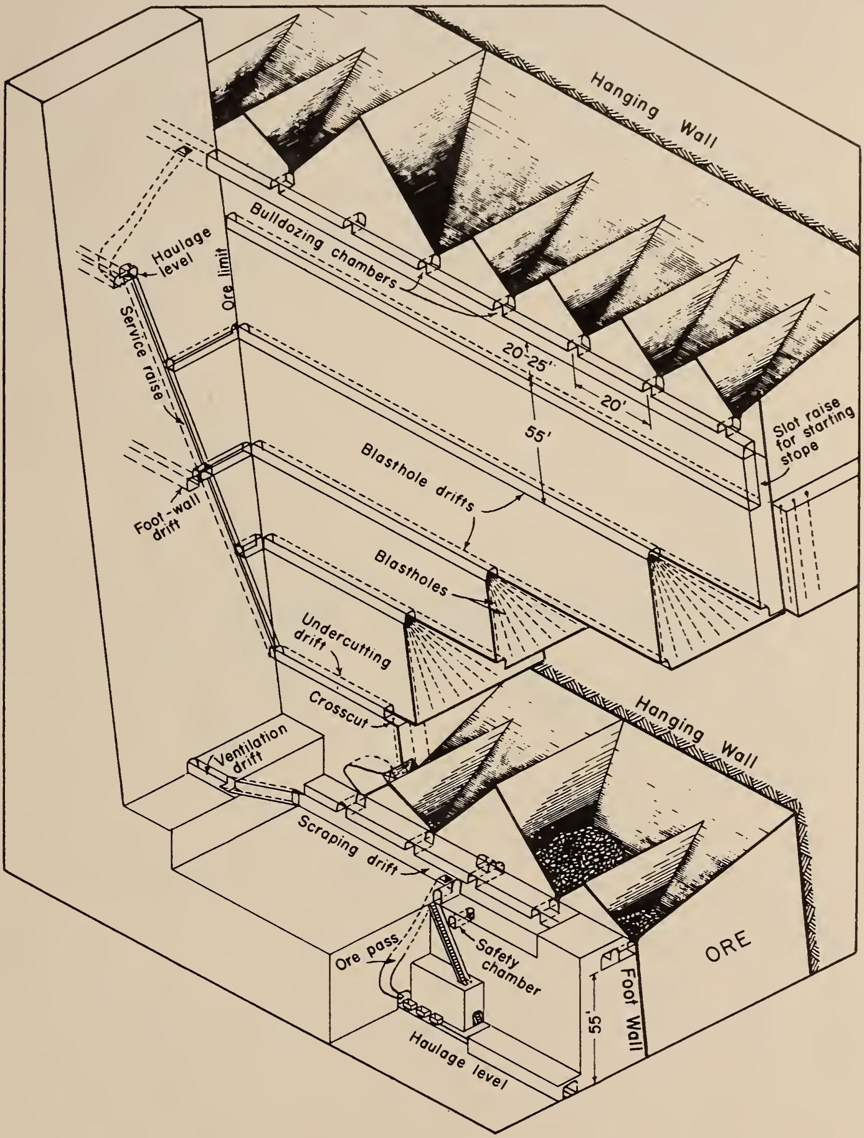
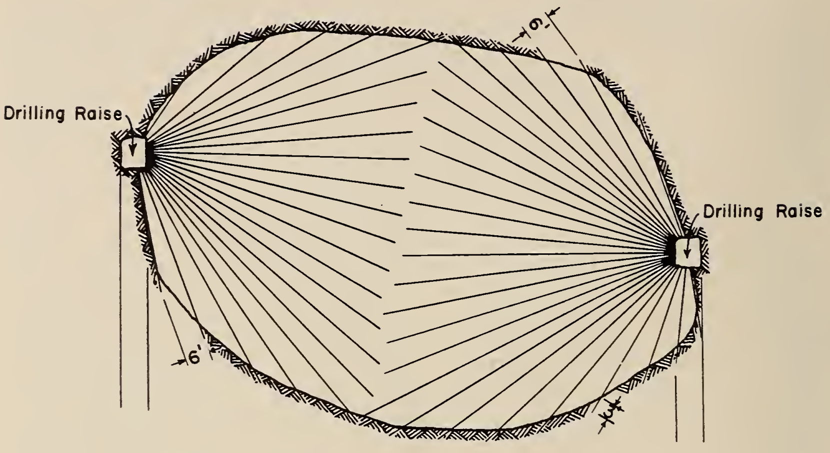
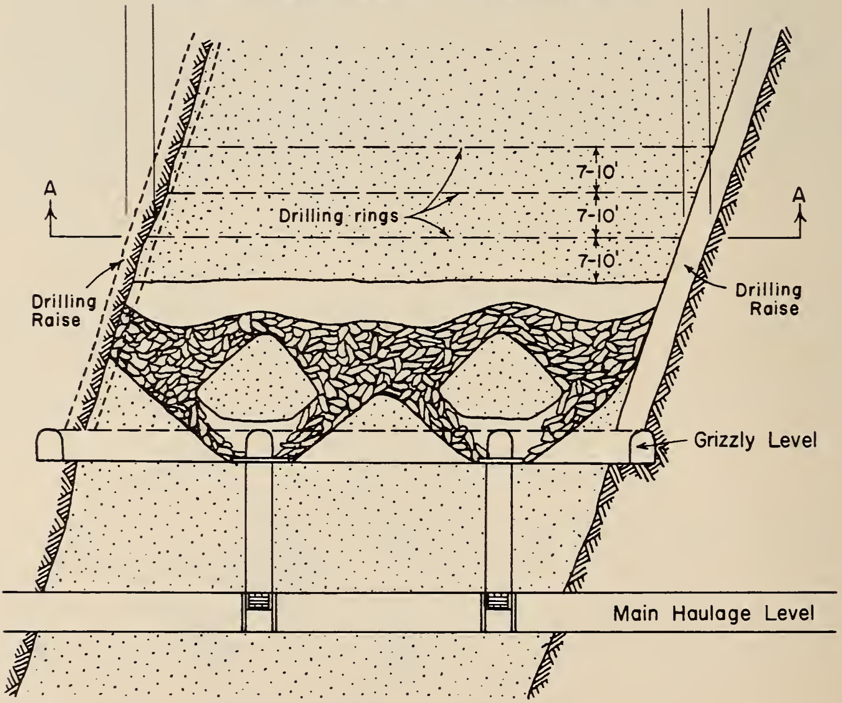


FIGURE 26.—Radial Blasthole End Slicing, as Shown in Isometric Drawing. Has Successfully Replaced the Bench or Aldermac Method.



Horizontal Section A-A showing drilling pattern



Vertical Section

Figure 27.—Horizontal Radial Blasthole Shrinkage Stoping.

Lake Superior district given in the Proceedings of the Lake Superior Mines Safety Council for 1949 to 1953. Table 7 shows the accidents from all causes associated with three caving methods, and table 8 shows those from falls of ground only. Of the three methods, top slicing has the highest severity and fatal rates. In this connection, it is noted that top-slicing methods have not been materially changed since the period 1938–42 when it had the distinction of being the safest of the underground mining methods. Top slicing in the Lake Superior district at present is only used where necessary to mine small ore bodies that cannot be mined by block-caving or by sublevel-caving methods. A comparison of man-hours' work performed in top slicing in the Lake Superior district in 1942 and 1953 show a decline of 86.9 percent in 1953.

Fatal injuries from roof falls were about equally divided among the various mining methods. Nonfatal injuries also were about equally divided, with the exception of those occurring at open-stope mines where injuries occur at one-half to one-third the regularity of those at other mines. There is little difference between most methods as to the hazards of rock falls.

Accident rates from rock falls and slides of banks at opencut mines are about one-third to one-tenth of those in underground operations.

TABLE 5.—*Injury rates by mining methods, 1943–52*

[Compiled by the Branch of Accident Analysis, Division of Safety]

Method of mining	Average number of mines	Total number		Rate per million man-hours	
		Killed	Injured	Killed	Injured
Open stoping	125	221	12,883	1.00	58.06
Timbered stoping	36	116	7,011	1.00	60.33
Filled stoping	11	32	2,954	.86	79.45
Shrinkage stoping	31	69	4,794	1.69	117.59
Caving methods	33	111	8,009	.79	56.79
Combined methods	51	127	13,224	1.06	110.33
Opencut, power shovel or dragline	162	120	5,918	.33	16.43
Opencut, hand loading	3	2	137	.96	66.08
Total	452	798	54,930	.77	52.86

TABLE 6.—*Rock-fall injury rates, by mining methods, 1943–52*

[Compiled by the Branch of Accident Analysis, Division of Safety]

Method of mining	Number killed	Number injured	Rate per million man-hours		Percent of total	
			Killed	Injured	Killed	Injured
Open stoping	93	1,454	0.42	6.55	42	11
Timbered stoping	52	1,633	.45	14.05	45	23
Filled stoping	18	611	.48	16.43	56	21
Shrinkage stoping	17	642	.42	15.75	25	13
Caving methods	50	1,716	.35	12.17	45	21
Combined methods	43	2,078	.36	17.34	34	16
Opencut, power shovel or draglines	25	309	.07	.86	21	5
Opencut, hand loading	----------	21	--------	10.13	--------	15
Total	298	8,464	.29	8.15	--------	----------

TABLE 7.—*Injury rates by caving methods, Lake Superior district, 1949–53* [1]

Method of mining	Man-hours worked	Total number		Man-days lost or charged to injuries	Rate per million man-hours		Severity rate per thousand man-hours
		Killed or totally disabled	Injured		Killed or total disabled	Injured	
Sublevel caving	51, 136, 064	25	1, 303	212, 201	0. 49	25. 48	4. 15
Top slicing	9, 470, 433	6	305	51, 561	. 63	32. 21	5. 44
Block caving	6, 779, 588	3	326	28, 774	. 44	48. 09	4. 24
Total	67, 386, 085	34	1, 934	292, 536	. 51	28. 70	4. 34

[1] Compiled from the Proceedings of the Lake Superior Mines Safety Council.

TABLE 8.—*Falls of ground injury rates, by caving methods, Lake Superior district, 1949–53* [1]

Method of mining	Killed or totally disabled	Number of injuries	Man-days charged to injuries	Rate per million man-hours		Severity rate per thousand man-hours	Percent of total		
				Killed or total disabled	Injured		Killed or total disabled	Injured	Man-days charged
Sublevel caving	11	147	73, 885	0. 22	2. 87	1. 44	44. 0	11. 3	34. 8
Top slicing	2	37	16, 007	. 21	3. 91	1. 69	33. 3	12. 1	31. 1
Block caving	0	33	735	--------	4. 87	. 11	--------	10. 1	2. 6
Total	13	217	90, 627	. 19	3. 22	1. 34	38. 2	11. 2	31. 0

[1] Compiled from the Proceedings of the Lake Superior Mines Safety Council.

OPENCUT METHODS

Opencut or open-pit methods have a wide application in mining ore deposits at or near the surface. If the top of the deposit is below the surface, the overburden and barren capping must be removed in advance of the open-pit mining. The removal of this top material is known as stripping. The stripping-pit limits must be extended beyond the limits of the ore pit to provide a protecting bench, and the slope of the pit walls must be such as to prevent sloughing and sliding of overburden into the pit area (fig. 28). As the ratio of stripping to ore made available for opencut mining increases, the cost of stripping per ton of ore increases until it reaches the economic limit for opencut mining. This limit is generally at the point where the cost exceeds that of underground mining.

There are two general types of opencut mining; in the first the waste material is removed to a location outside the pit, and in the second the waste is placed in the part of the pit from which ore has been mined, after a preliminary cut has been made. The second plan can be followed where the ore deposit can be mined to the bottom at some point, and the excavation can be continued horizontally in such a way that haulage can be maintained.

Where mining must proceed in successive vertical stages, the stripping area must often be enlarged as greater depths are reached.

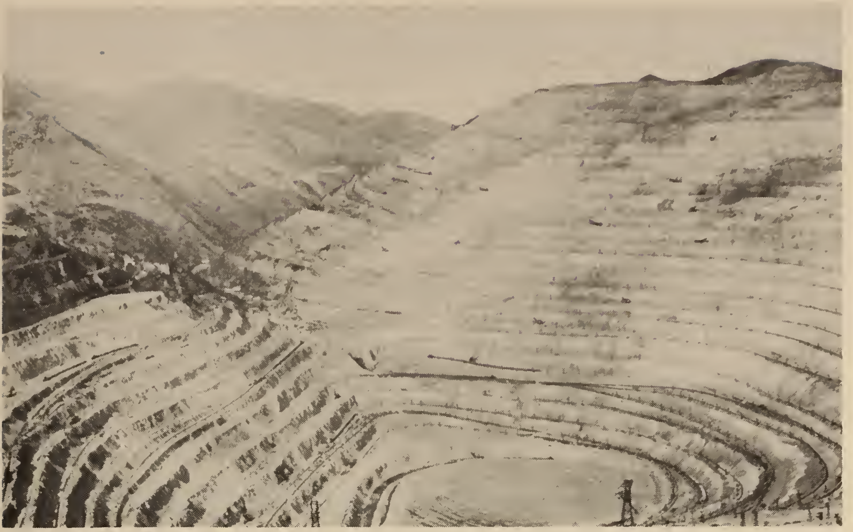


FIGURE 28.—Opencut Mine.

The safe slope angle that must be maintained will be governed by the character of the overburden or capping, the degree of water saturation, and the direction of bedding planes, faults, and other breaks in the formations. The height of benches and depth of pit may also affect the safe maximum slope. The slope of each bench can be varied to suit local conditions, but the overall average of the sides of the pits must be determined to prevent dangerous or extensive slides. Berms left in the face of a pit as each bench is finished reduce the overall slope of the sides and hold spalls from weathering and chunks loosened by frost and rain. As the pits are relatively deeper, more attention has been paid to the angle of slope at copper mines than at iron-ore properties. At many iron-ore mines the overall slope is the same as the individual bank slope.

To avoid danger from slides or falling rocks, it has been recommended that benches in blocky ore in an opencut mine should not be higher than can be trimmed by the power shovel used.[21] Time is an important factor of ground movement in high banks. Rain, snow, freezing, and thawing hasten ground movement; as time passes the movement gradually accelerates until a slide occurs. Minor dislocations of rock usually serve as a warning of a major slide. When trouble is anticipated on steep fractured banks, benchmarks are usually established at critical points on the bank. The points are closely watched and surveyed at intervals; thus, major slides can be anticipated. Banks may be carried at a relatively steep angle when mining can be completed in a short time; the longer the banks stand the more dangerous they become.

Hand labor, railroad- and tractor-mounted steam, diesel, and electric shovels, and draglines are used in opencut procedures. Hydraulic and dredge operations and glory-hole mining are also types of opencut or surface mining but are not discussed in this publication.

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[21] Gardner, E. D., and Mosier, McHenry, Opencut Metal Mining: Bureau of Mines Bull. 433, 1941, p. 106.

Advantages.—Opencut methods have the great advantage of being conducted largely in daylight; at night floodlights may be used advantageously. Mechanical methods of loading and haulage are easily applied, resulting in small labor requirements per ton of ore produced and reducing the time of exposure of workers for a given tonnage. A system of operations can be used that can be closely supervised. The safety records of opencut operations are generally much better than for underground methods. Mining costs per ton, including development and upkeep, are relatively low, and the rate of production can be varied greatly; maximum productions per man per hour or per man per year can usually be achieved by opencut methods.

Disadvantages.—Considerable expense and work are often incurred in preliminary stripping. In severe weather, opencut work has manifest disadvantages, such as freezing of air and water lines, banks and blasted material that must be moved, slippery and softened haulage roads, clearing of snow, loosened and unsafe banks, and exposure of men. There is often necessity for preventing flooding of the pits during heavy rains and spring thaws. In deep pits a difficult problem often is presented in keeping the proper slopes and benches on the sides of the pit to prevent slides. Deep pits must also have enough horizontal extent to permit reasonable haulage grades, or substitute methods must be adopted for bringing the ore to surface level.

PREVENTION OF ACCIDENTS FROM ROCK FALLS OR SLIDES

Accidents in opencut mining may be classed,[22] [23] as: Slides of bank, blasting accidents, accidents around shovels, drills, or other equipment, falls of persons, accidents from haulage, and shop accidents. Judging from company and State reports, blasting accidents and haulage accidents are the most frequent.

Slides or falls of rock cannot be considered comparable with falls of ore or rock underground as a source of injury. For much of the time the bank is clearly visible, and threatening conditions are fairly readily discerned. A slide of bank usually gives some warning so that escape may be possible even after the slide starts. Avoidance of the danger from slides generally consists of keeping the banks as flat as is economical, watching them carefully, and avoiding work directly under very high banks during inclement weather and during thaws.

Where shovels are operated, men should not work between the shovel and the bank more than necessary, so that in case of a rush of material they would not be crushed against the shovel or be penned in by it. For night work bright illumination should be employed.

Small quantities of rocks or even single rocks may fall from a high bank with enough force to kill or seriously injure anybody that they may strike. Where this danger is known to exist, men should be employed whose sole work is to trim the loose material off the banks. The use of large equipment able to handle large chunks greatly reduces the need for blockholing and the resultant exposure of men under unsafe banks. The use of large-diameter drills also permit placing blastholes a greater distance from the crest of the bench banks, thereby reducing the hazard of men and equipment being carried down with a fall of bank.

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[22] Cash, F. E., and von Bernewitz, M. W., Methods, Costs, and Safety in Stripping and Mining Coal, Copper Ore, Iron Ore, Bauxite, and Pebble Phosphate: Bureau of Mines Bull. 298, 1929, p. 255.

[23] Harrington, D., Worcester, A. W., and East, J. H., Jr., Information on the Prevention of Quarry Accidents: Bureau of Mines Bull. 473, 1950, p. 4.

Scaling quarry faces by hand,[24] especially high faces, is a hazardous operation even when safety belts and lines are provided to protect scalers. In steep-bedded deposits, fracturing parallel to the face may prevent adequate scaling, and slides of rock may occur after all apparent loose rock has been scaled down. When inspection of the top of the bank above the face discloses such slips or fractures, the material should be blasted down before men are allowed to work under the face. Vertical walls are recommended in relatively flat-bedded deposits, but they are generally hazardous in steep-bedded deposits. In every case the overburden should be stripped to bedrock at least 10 feet back from the face. Overburden should be sloped to its angle of repose.

Mechanical scaling by means of a shovel dipper is probably the safest method of scaling but is effective only where the height of the quarry face is within reach of the dipper. Mechanical scaling of high faces may be done by using a dragline bucket under certain conditions, but the terrain above the quarry face must be such that a dragline may be safely placed on the bank above the quarry face. In some instances the Industrial gun, using a 2-ounce ball, has been effectively used in shooting down loose rock. Hand-scaling methods have been improved by use of lightweight aluminum scaling bars and mechanical lift and suspension devices.

Quarry walls should be inspected and scaled, where necessary, of loose rock after every blast, rain, freeze, or thaw before men are allowed to work under the quarry faces.

Abandoned or inactive pits or quarries should be fenced or barricaded, and signs should be posted to warn persons of the danger of falling rock.

SUPPORT OF GROUND

The method of mining adopted must consider tha nature of the ground and assure safe support during the cycle of operations. Support can be accomplished in a number of ways.

A. Self or natural support.
B. Artificial support.
  1. Ore or waste.
  2. Internal support or reinforcement.
    (*a*) Freezing or grouting.
    (*b*) Rock bolts.
  3. External support.
    (*a*) Posts, stulls, or stringers.
    (*b*) Timber sets or cribbing, bulkheads.
    (*c*) Steel and concrete.

A. SELF OR NATURAL SUPPORT

In rare instances the nature of the ground and ore is such that it is practical to mine the ore from large, open stopes, or rooms without providing any support.

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[24] Work cited in footnote 23, p. 42.

In other cases, pillars of ore are allowed to remain in place to support the back or the walls. Should economic conditions warrant recovering these pillars, it is sometimes accomplished by retreating in that stope or by erecting temporary or provisional support of timber, rock bolts, steel cables, or concrete.

Recovering pillars is generally classed as hazardous work in comparison to the original or initial operation.

Where pillars are left to provide natural support they may become weakened through slow crushing or from spalling through weathering. These pillars may be reinforced with cable and by rock bolting. To reduce weathering action, they are sometimes gunited.

Arching the back of mine development drifts and crosscuts materially assists in self-support of the smaller mine openings.

B. ARTIFICIAL SUPPORT

1. ORE OR WASTE

A shrinkage stope is an example of walls supported by the broken ore in the stope. This support is maintained until all the ore in that stope section has been broken, after which the ore usually is drawn off as soon as possible to avoid dilution by waste caving or falling off the walls.

Cut-and-fill stopes typify support of ground by using waste. Such stopes may be either flat-back or rill-type, and frequently no auxiliary timber is used. Waste from development workings or from "waste" stopes is brought to the stope on the level above. Rock walls, often called "pack walls," have been constructed to support mine workings since the earliest ages. Figure 29 shows an example of a rock dry wall for support of roof. Tailings from the concentration of ore have been employed advantageously in a number of mines. Use of this material for fill has several advantages, especially where enough waste for stope filling is not broken in current development openings (fig. 30).

2. INTERNAL SUPPORT OR REINFORCEMENT

(a) **Freezing or Grouting.**—When ground to be mined is not self-supporting in the opening desired, there exists the possibility of internal reinforcement, either temporary as in freezing the ground until a permanent support can be applied, grouting with various cements, rock bolting, or a combination of grouting and rock bolting. All have been known and have been used occasionally for many years, particularly in emergency support when cost was not a factor to be considered. However, this type of support can now be considered for economical production mining because of recent improvement in the development of new grouting mixtures and grouting techniques, improvement in drill bits and drilling techniques, and improvement in mechanized mining methods, which put a premium on available working space.

At least three manufacturing companies have developed quick-setting cements (one develops 50 percent of its ultimate strength within 30 seconds after application) in which contraction after setting is eliminated and penetration into many rocks is obtained. At present these cements are being used in mining for filling voids that have developed from decay of the original piling between concrete

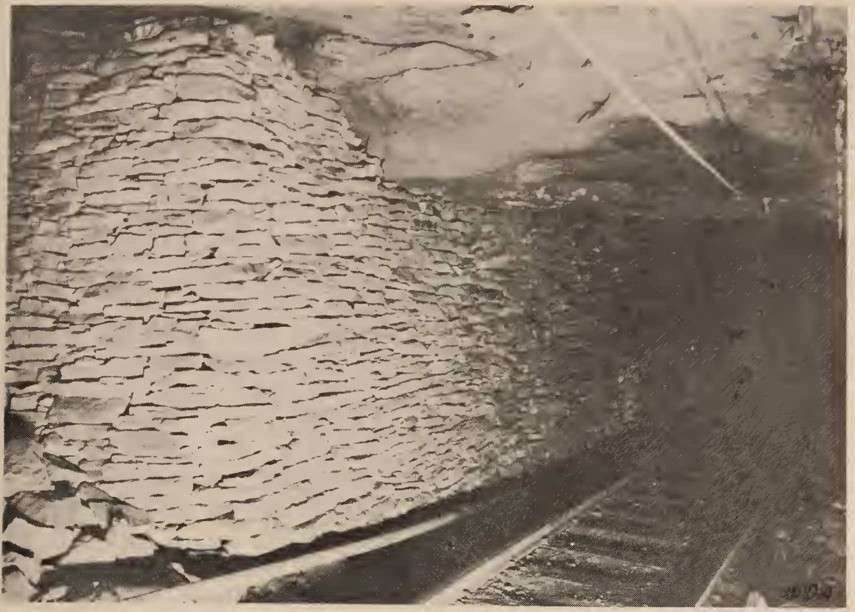

Figure 29.—Dry-Wall Support for Roof.

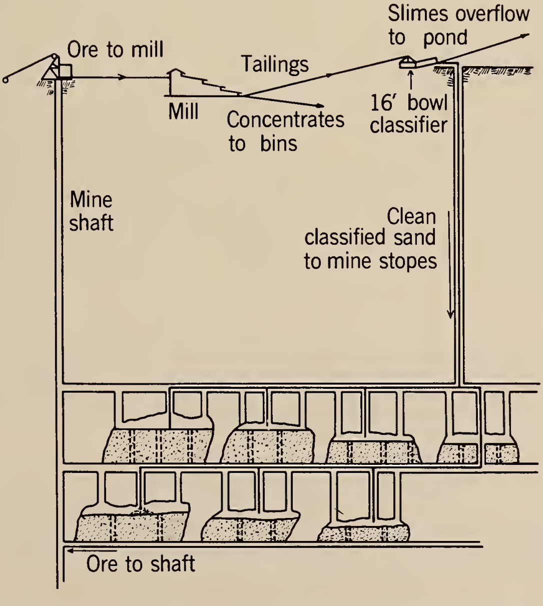

Figure 30.—Flowsheet Showing System of Filling Stopes With Mill Tailings.

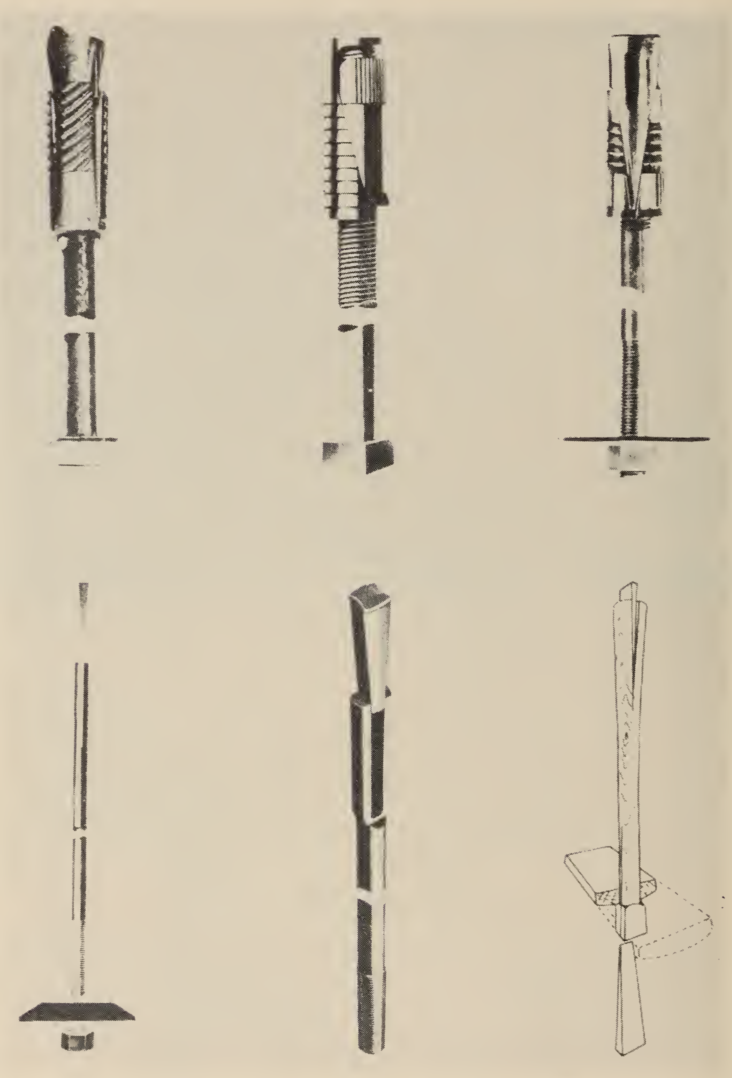

FIGURE 31.—Typical Bolts in Common Use Designed Primarily to Clamp Strata Through Maintenance of Tension Between an Anchoring Device and a Bearing Plate at the Collar of a Drilled Hole. Three Varieties of Expansion Shell Bolts, Two Varieties of Slotted Steel Bolts, and the Wooden Bolt Developed by Lanier Are Shown.

tunnel linings and solid rock and to consolidate water-bearing rocks preparatory to mining, eliminating the need for caisson work.

(b) **Rock Bolting.**—The basic difference between bolting and conventional support is sometimes overlooked. Successful bolting depends upon being able to make the ground itself an integral part of the support structure (figs. 31 and 32), whereas with conventional supports one assumes that failure of the ground is inevitable and prepares to support the ground within certain limits beyond which the ground is considered to be self-supporting.

In the light of experience that has been accumulated, from the standpoint of utilization by the engineer the following classifications of bolting appear to be logical:

1. Suspension: Pinning a loose piece of rock to ground considered self-supporting.
2. Beam building: Using bolts to cause a series of bolts with little or no bond between them to act as a monolith. By reducing this action of the beds to that of a monolith stresses are reduced to a value which the loads can safely resist. This type of bolting is used principally in rectangular openings.
3. Reinforcement of the skin area of an arched opening to provide additional support to counteract the effects of tensile, compressive, and shear stresses.
4. Reinforcing walls against stresses that are totally or predominantly shear and compression.

Suspension bolting.—This type of bolt was the prototype of modern rock bolting. At the St. Joseph Lead Co. mines, which deserve a major portion of the credit for pioneer work in bolting, this type of bolt is called a "shinplaster" and is used with no particular pattern to fasten single pieces of loose or questionable material in areas that are otherwise considered self-supporting. Such bolts should, of course, be strong and have adequate anchorage to support the estimated weight of the ground under question, with a factor of safety of at least 3 to compensate for uncertainties of anchorage and load.

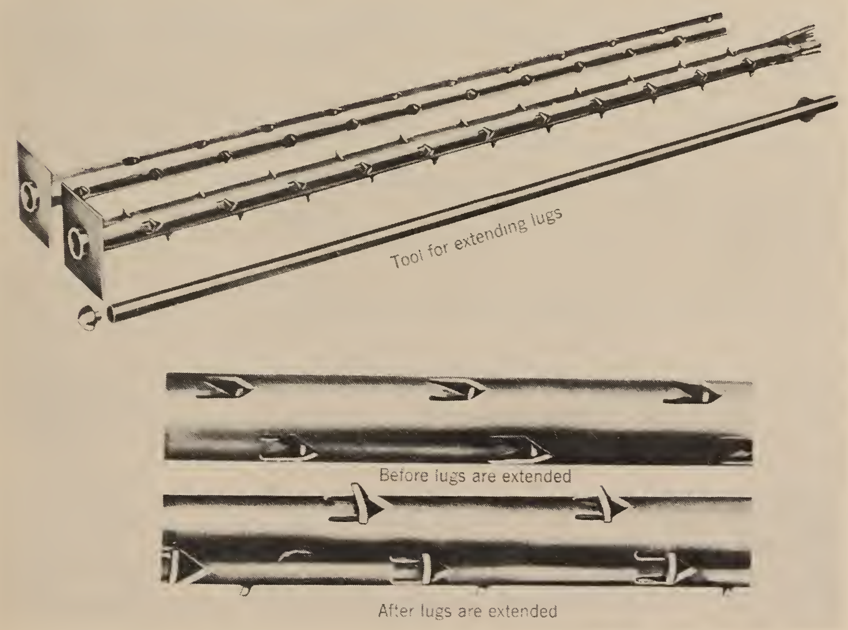

FIGURE 32.—Bolts Designed to Afford Resistance to Shear Stresses.

Beam building.—Two aspects of this class of bolting should be considered: (1) Roof bolts may be used in stratified and loosely consolidated rocks to bind them together so that collectively they act as a single beam rather than a series of separate beams; (2) with the bolts that now are considered conventional (slotted type and expansion shell type), an undesirable disturbance of the strata is brought about by the action of the bolt at its anchorage. This is due to exertion of a tremendous thrust against the rock forming the side of the drill hole, which is required to anchor these bolts properly (so the anchorage will develop the full strength of the bolt). This thrust acts normal to the long axis of the drill hole and tends to create a fissure in the rock at the place of anchorage, extending parallel to the stratification that normally would also be parallel with the roof surface of the passageway supported. In pattern bolting these fissures from the individual bolts connect and form a continuous weak horizon segregating the monolith, formed by the bolting, from the rock above.

Therefore, with these bolts, it is important that no dependence should be placed upon the hanging or "suspension" of a weak shale by drilling a few inches into a massive and strong formation above; the bolts should be long enough so that the total thickness of the resulting monolith renders the beam self-sufficient.

It is believed that over 80 percent of the current bolting being done, that is, that being done in horizontally bedded mineral deposits, is in the category of beam building.

Figures 33 and 34 are typical of successful beam-building bolting methods. Wooden bolts are used in a few coal mines in Kentucky and Indiana to consolidate roof beams with and without supplementary wooden props. The application of wooden bolts for this purpose appears to be limited to marginal support problems. However, where they can be used successfully the material costs for wooden bolts are approximately 50 percent cheaper than for steel bolts.

In connection with cut-and-fill stoping rock bolts are used at several mines in the Porcupine gold-mining field in Ontario for temporary support of the back in almost vertical pitching veins wherever it is inclined to spall (fig. 35). These are installed at the beginning of the mucking cycle to support the back and are blasted out when the next cut is taken.

An ingenious method of roof support is practiced at A/S Sulitjelma Gruber mine at Sulitjelma Gruber in Norway.[25] The ore is 1 to 2 meters thick and where it dips 10° to 30° is mined by scrapers working on a long face which closely resembles longwall coal mining. Conventional rock bolts with cribbed pillars of schist blocks are ordinarily used, but where extra roof support is required ½-inch wire rope is "sewed" and grouted into the vertical drilled holes as shown in figure 36.

Reinforcement of skin area of an arched opening.—General mining experience has shown that a mine opening having an arched roof or back is less likely to require artificial support than a rectangular opening in the same material. Moreover, this practice indicates that the circular or semicircular opening seems to be the most desirable.

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[25] Austin, C. C., Roof Sewing: Eng. and Min. Jour., vol. 154, September 1953, p. 91.

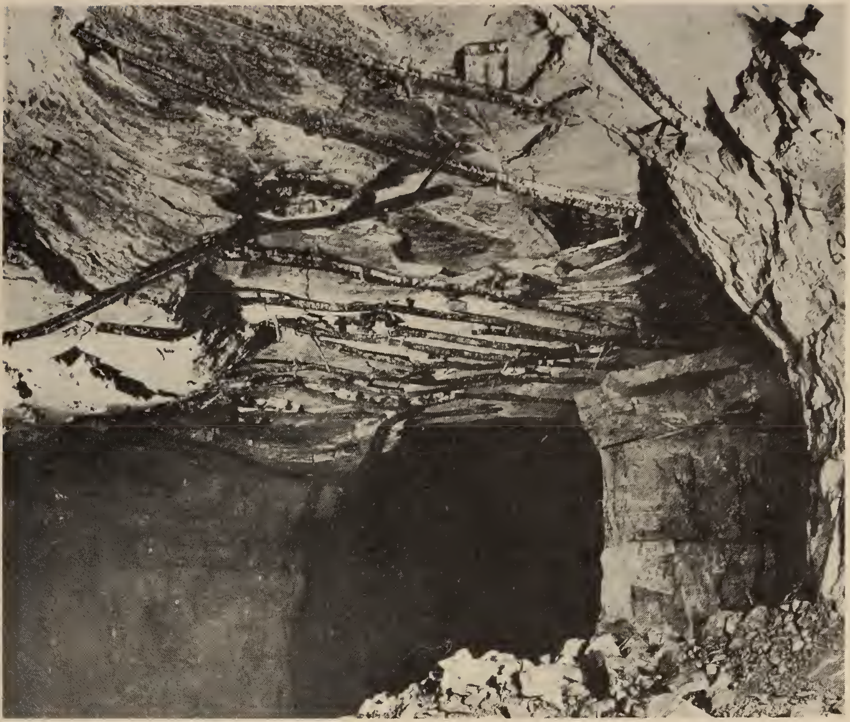

FIGURE 33.—Pattern Bolting With 8-Foot, Slotted-Type, Angled Bolts Using 4-Inch Channels as Bearing Plates to Thinly Bedded Limestone Back. (Courtesy, St. Joseph Lead Co.)

The necessity for this use of the roof bolt comes from two causes, namely: (1) The nature of the stresses about mine openings (in this connection it should be kept in mind that rocks are very weak and unreliable in tension); (2) the irregular shape of an opening made by conventional tunnel driving methods and the fracturing and shattering in the sides of an opening caused by blasting incidental to tunnel driving.

When any mine opening is made the most intense stress is encountered within a short distance of or at the skin of the opening. The effects of the opening in rock stress, however, decrease very rapidly with the distance into the ribs or back of the opening. It is safe to assume that at a distance into the back or ribs equal to the long cross-sectional dimension of the opening the increased concentration of stress caused by the opening is negligible.

Whether or not the skin stress will be tensile or compressive is determined by the degree of arching and the ratio of the width of the opening and that of the vertical load on the opening, as well as the orientations of the principal axes of the opening to the direction of the acting load. Of these factors the magnitude of the horizontal load on the opening is almost always unknown. Consequently, it is never certain that the back or roof of the arched opening will not be in tension.

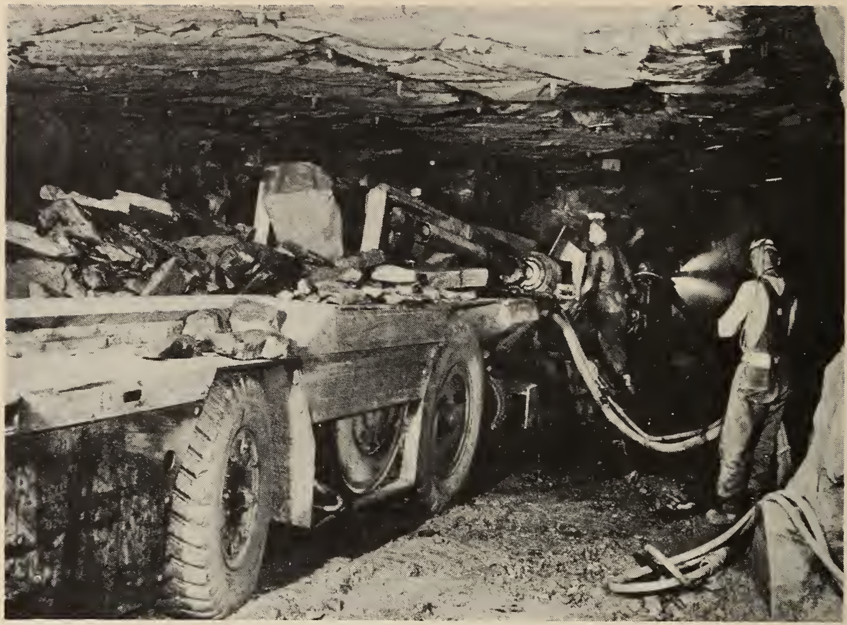

FIGURE 34.—Pattern Bolting Using 6-Foot, Slotted-Type Bolts on 6-Foot Centers (Laterally and Longitudinally to Direction of Roadways) to Support a Thinly Bedded Shale Roof. (Courtesy, Tennessee Coal and Iron Div., United States Steel Corp.)

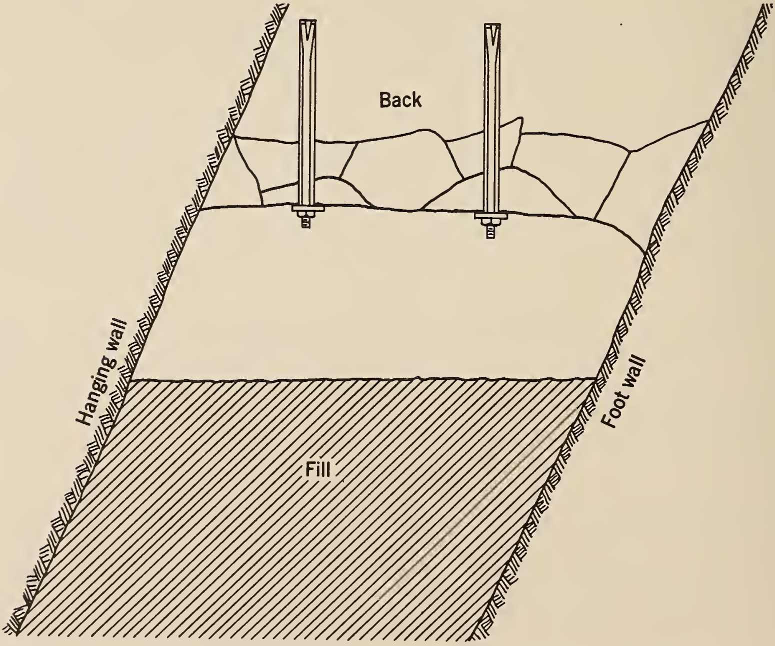

FIGURE 35.—Support of Weak Back in Cut-and-Fill Stoping.

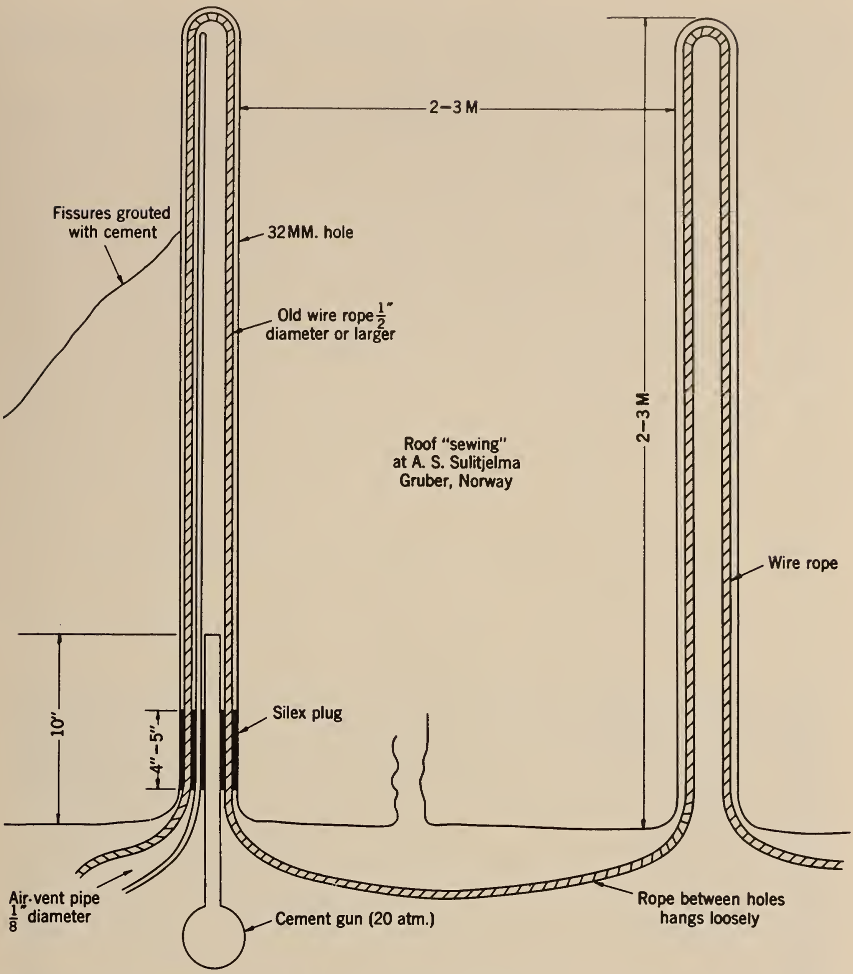
2–3 M
Fissures grouted with cement
32MM. hole
Old wire rope $\frac{1}{2}$" diameter or larger
2–3 M
Roof "sewing" at A. S. Sulitjelma Gruber, Norway
Wire rope
10"
4"–5"
Silex plug
Air-vent pipe $\frac{1}{8}$" diameter
Cement gun (20 atm.)
Rope between holes hangs loosely

Figure 36.—Steps in Roof Sewing: (1) Drill Hole 2 to 3 Meters Deep, (2) Insert Wire Rope, (3) Insert Vent Pipe, (4) Insert Short Pipe Connected to Grouting Pump, (5) Insert Silex Plug, and (6) Grout Hole with concrete gun at 300-pound pressure.

(Courtesy, Engineering and Mining Journal.)

Even where the roof is in tension, it is certain that within a relatively short distance of the surface the rock is in compression, or at worst under very low tension. Relatively short rods, rarely not longer than 6 or 8 feet, will reach rock to which the bolts can be anchored to suspend any ground that may become loose due to rupture from tension in the back.

Tensile stresses, moreover, are not necessarily the only offenders in causing ground fracture in the backs of arched openings. Compressive stresses, if sufficiently intense, may cause flaking and sloughing of ground from the skin. Again, it should be kept in mind that the intensity of these stresses decreases rapidly as the distance from the skin into the rock increases. Therefore, systematic bolting may be used to compact major pieces of rock that tend to separate from the skin because of compressive stress.

In tunneling, if an opening is being made in rock that has enough inherent strength to withstand the load of the superimposed strata and if it were possible to cut exactly to a semicircular *A* line, no artificial support would be required (see fig. 37). However, from a practical standpoint where explosives are used this is not possible; the blasting produces a jagged breakline, exposing and opening bedding or jointing planes, and produces fractures that extend into the back and ribs. Therefore, the actual archline must be considered to be outside of the maximum overbreak or *B* line, and the rock between *A* line and *B* line may require artificial support. The radius of this *B* line depends upon the effect of the explosives used on bedding or jointing planes, but a slotted-type bolt whose penetration is the difference in radius between *A* line and the deepest penetration of the overbreak multiplied by two appears to offer a satisfactory factor of safety.[26] This is borne out by the published information on large-bore tunnel driving (fig. 38) and in the drift and crosscut experience of the Anaconda Copper Co. in its Butte, Mont., mines.[27]

When sloughing or spalling occurs about the skin surface of an arched opening, owing either to excessive skin stress or to weathering, bolting has been found to have definite value in compacting and reinforcing this skin area. Although the conventional slotted-type bolt was used successfully for this purpose in the Bureau of Reclamation's Duchesne tunnel in Colorado, it is believed that, inasmuch as ordinarily the only possibility is to place bolts normal or nearly normal to the skin surface, a bolt that fills the entire drilled hole would be more efficient. This appears to be the best application that has been found thus far for the wooden bolt [28] developed by Sterling Lanier, Jr. This bolt is used in the Day mines (fig. 39) in the Coeur d'Alene district to stabilize the skin area of arched drift openings in a loosely consolidated and shaly quartzite. It would not be feasible to attempt to anchor a steel bolt in this rock using any of the fastening devices now on the market; moreover, about two-thirds of the bolt length bears against the circumference of the drill hole.

At Lake Shore and Macassa mines [29] [30] in the Kirkland Lake gold-mining district in Ontario, rock bolts are utilized to combat the hazard of rock bursts for the following reasons:

1. To compact the skin area of arched openings attempting to supply enough additional reinforcement so that the openings are not the weakest portions in a mass of ground being subjected to extraordinary loads that may induce rock bursts (fig. 40).
2. Where, because of past experience, it is felt that it is inevitable that a rock burst will occur during the active life of a drift or crosscut, bolts are used to support a cushioning structure intended to contain the flying rock (fig. 41).

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[26] Platt, D. H., Roof-Bolting the Delaware Aqueduct: Bureau of Mines Inf. Circ. 7652, 1952, 9 pp.

[27] O'Leary, V. D., and Pollish, Lloyd, Rock-Bolt Developments at Butte: Mining Sec., Nat. Safety Congress and Exposition, Chicago, 1953.

[28] Farmin, Rollin, and Sparks, Carville E., The Use of Wooden Rock Bolts in the Day Mines: Trans. AIME Min. Branch, vol. 196, pp. 922–924 (Tech. Pub. 355A); Min. Eng., vol.5, September 1953, pp. 922–924.

[29] Robeson, W. T., Rock-Burst Incidence, Research and Control Measures at Lake Shore Mines, Ltd.: Trans. Canadian Inst. Min. and Met., vol. 49, 1946, pp. 347–374.

[30] Seymour, Morris, Rock-Bolting Practices in Canadian Metal Mines. IV. At Lake Shore Mines: Mine Accident Prevention Assoc. of Ontario Symposium, Toronto, May 1953. Canadian Min. Jour., vol. 74, No. 9, September 1953, pp. 62–65.

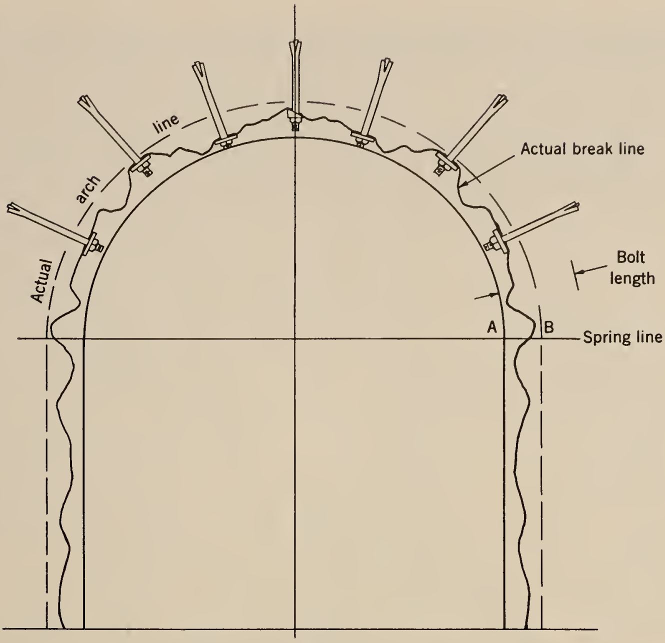

FIGURE 37.—Bolting Applied to Tunneling.

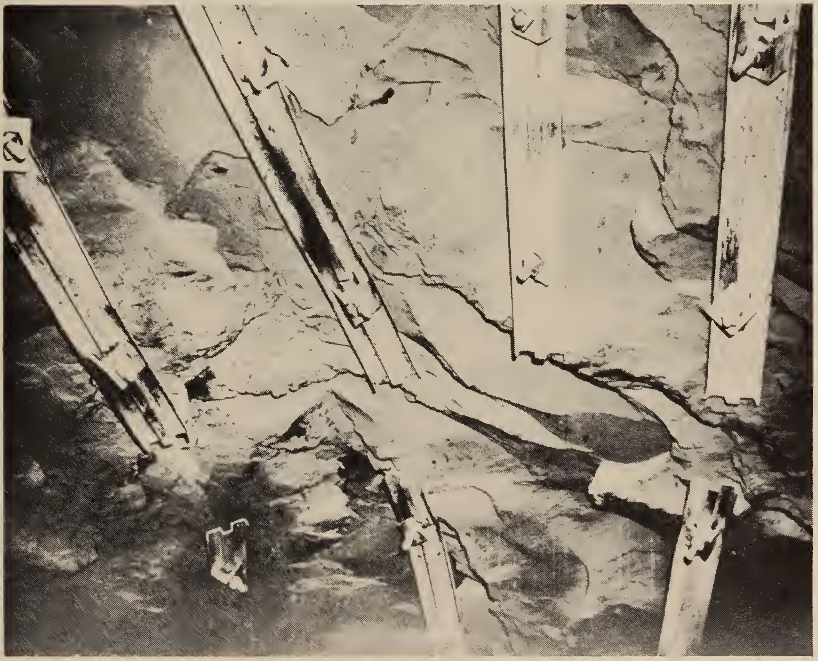

FIGURE 38.—Bolting in the Delaware Aqueduct Tunnels. East Delaware Tunnel, Suspension-Roof-Support Specifications.
(Courtesy, New York Board of Water Supply.)

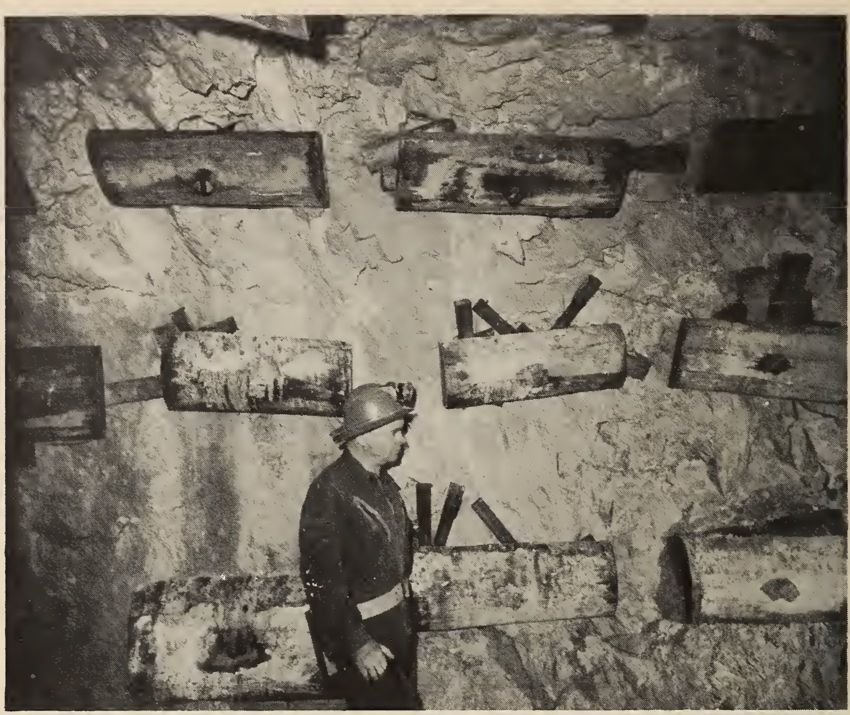

FIGURE 39.—Wooden Bolts Used to Compact Arched Openings in Loosely Consolidated and Shaly Quartzite.
(Courtesy, Day Mines, Inc.)

Reinforcing walls against loads that may be totally or predominantly shear and compression.—The chief reason for not including this heading in the previous category is to emphasize that in shafts or steeply pitching stopes there exists the probability of resisting a load resulting from hundreds of feet of an unsupported vertical span, regardless of whether these walls are sedimentary rocks or the contact-zone rock of an igneous formation. A bolt freely suspended between an anchoring device and a bearing plate contributing only tension to the structure and in a direction normal to application of the load appears not to be the most efficient device to use for compacting such a skin area. What is needed is a bolt that will better constrain motion in a vertical direction. A conventional bolt will resist shear stresses after the ground starts to move, but then the load can no longer be considered static and the ground has started to break up. Failures of this type have been experienced in both the United States and Canada.

Nevertheless, where the unsupported span is not too high, the effect of strain resulting from the vertical component of this column has been combated effectively with the ordinary slotted-type bolt as at Butte.[31]

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[31] Work cited in footnote 27, p. 52.

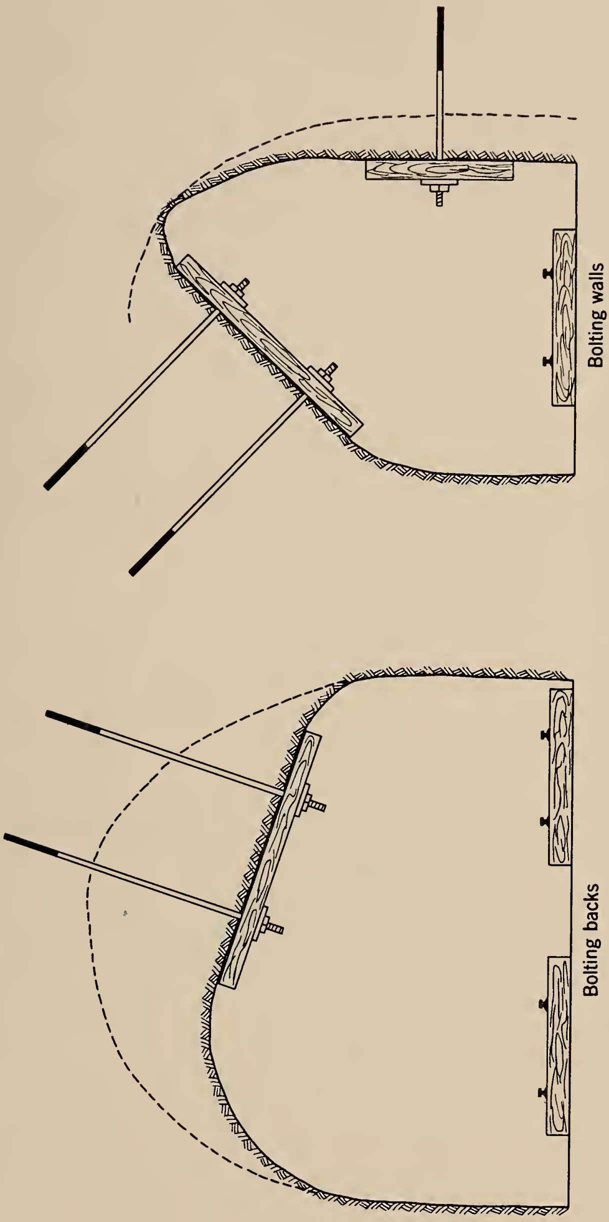
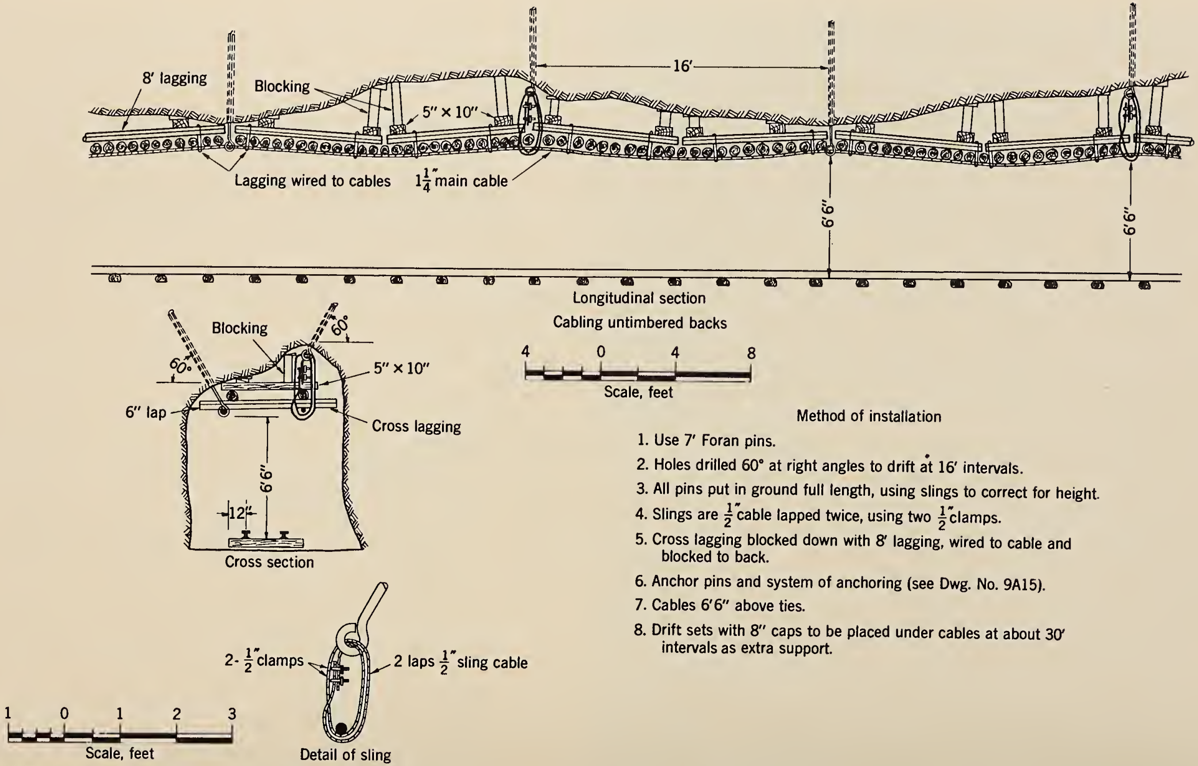


FIGURE 40.—Rock Bolts Used to Compact Portions of the Skin Area in Anticipation of Extraordinary Loads That May Induce Rock Bursts.
(Courtesy, Lake Shore Mines, Ltd.)



Method of installation

1. Use 7′ Foran pins.
2. Holes drilled 60° at right angles to drift at 16′ intervals.
3. All pins put in ground full length, using slings to correct for height.
4. Slings are $\frac{1}{2}$″ cable lapped twice, using two $\frac{1}{2}$″ clamps.
5. Cross lagging blocked down with 8′ lagging, wired to cable and blocked to back.
6. Anchor pins and system of anchoring (see Dwg. No. 9A15).
7. Cables 6′6″ above ties.
8. Drift sets with 8″ caps to be placed under cables at about 30′ intervals as extra support.

FIGURE 41.—Rock Bolts Used to Support a Cushioning Structure to Contain Flying Rock so as to Prevent Injury to Personnel.
(Courtesy, Lake Shore Mines, Ltd.)

Virtually all the veins in Butte have a common fault, which is a badly jointed hanging wall. This false "hanging wall," as miners call it, is from 1 to 6 feet thick and makes it almost impossible to do mining without timber support. Horizontal cut-and-fills without timbers were tried; but, due to poor waste supply and slowness of filling, a great deal of trouble was caused by waste sloughing from the walls into the ore stream, causing dilution and in some instances loss of the ore block.

A study of the hanging wall indicated that the heavy blocks of ground moved on the joints in a sliding, downward fashion. This movement set up a shearing stress along the contact so that the ore would be dragged down and eventually fall. If the sliding movement in the hanging wall could be temporarily stopped until fill was placed against it, it is possible that stopes could be operated without timber. Rock bolts were the practical answer to this supposition (fig. 42).

The slotted-type bolt is being used successfully in several other ore-mining districts in the United States and Canada to compact the walls of cut-and-fill stopes where the vertical distance between the back and the fill is not excessive.

In 1951–52 a 11-foot-square vertical air shaft was sunk 219 feet through sedimentary rocks at the Stansbury mine of the Union Pacific Coal Co., Rock Springs, Wyo., using 1-inch-diameter, 6-foot-long, slotted-type bolts (fig. 43) on 34-inch centers horizontally and 27-inch centers vertically to compact shale and sandstone walls. The bolts were installed over 12½-gage V-mesh wire 58 inches wide, with an overlap of 2 inches. The bolts and wire mesh were installed as the sinking proceeded; afterward it was decided to gunite the shaft to a minimum thickness of 1 inch outside the wire mesh.

The cost of bolting and installing wire mesh, including labor and material, was $35.52 per linear foot of shaft as compared to an estimated cost of $54.66 if the shaft was timbered with wooden timbers. The total cost, including the guniting, was $74 per foot of shaft as compared with an estimated cost per foot of $68.41 for a concrete lining. However, the estimate of the cost of the concrete lining does not include the indeterminate cost of temporary support that would be necessary preparatory to pouring the concrete.

3. EXTERNAL SUPPORT

Timber support in mine workings is used in many forms, ranging from a simple post to reinforced square sets or shaft timbering with squeeze sets (figs. 44 to 49). Posts, stulls, and stringers are applicable for local or limited areas requiring support. The value of these is greatly increased by the use of head and foot boards where a solid and substantial footing does not exist. Stringers in narrow stopes can be placed readily, give good support, and in many instances can be salvaged.

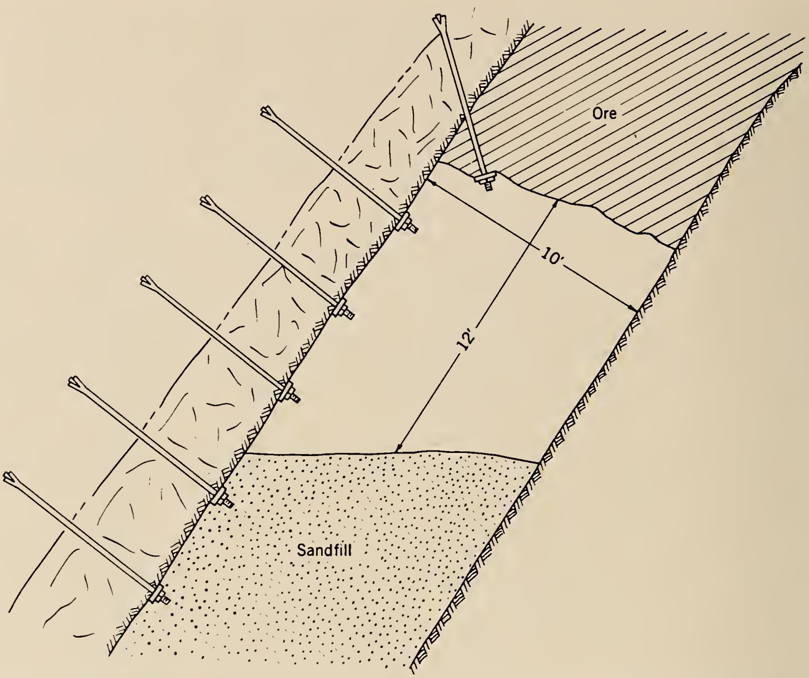


FIGURE 42.—Bolting Ore in Back to Rock in Hanging Wall While Fill is Placed. (Courtesy, Anaconda Copper Co.)

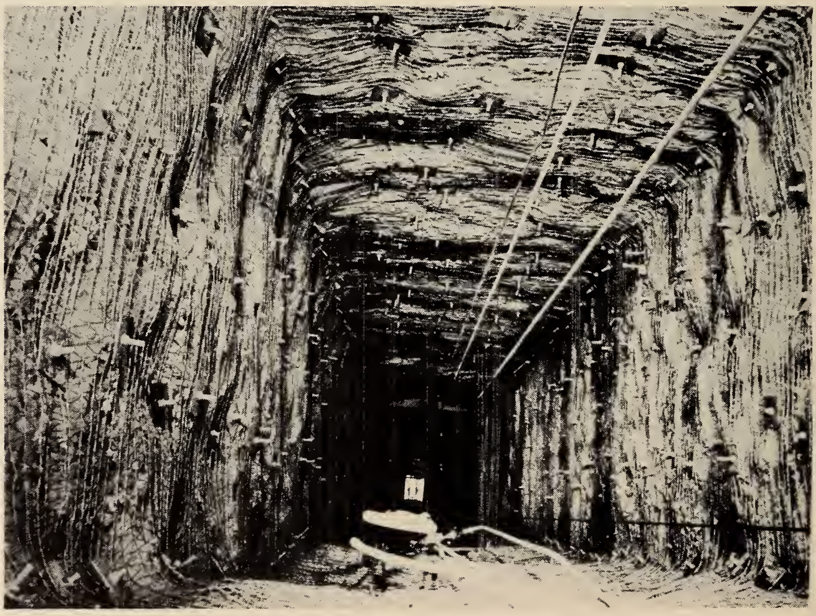
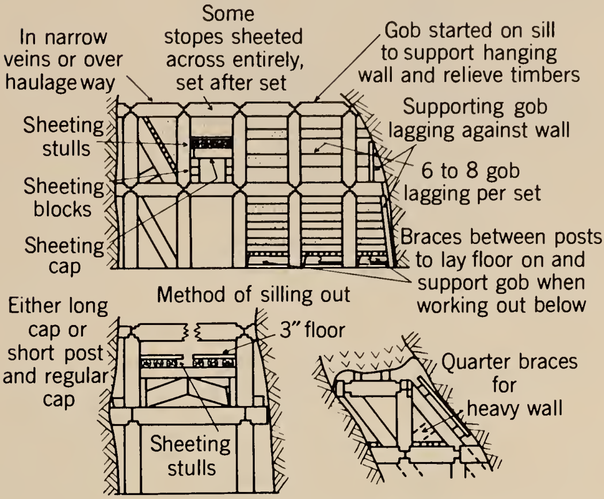


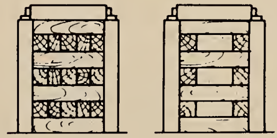
FIGURE 43.—Air Shaft With 6-Foot Rock Bolts Installed With Wire Mesh to Hold Shale and Sandstone Walls. (Courtesy, Union Pacific Coal Co.)


In narrow veins or over haulage way
Some stopes sheeted across entirely, set after set
Gob started on sill to support hanging wall and relieve timbers
Sheeting stulls
Supporting gob lagging against wall
Sheeting blocks
6 to 8 gob lagging per set
Sheeting cap
Braces between posts to lay floor on and support gob when working out below
Method of silling out
Either long cap or short post and regular cap
3" floor
Quarter braces for heavy wall
Sheeting stulls

Method of supporting filling over a wide set

Timbering in square-set raise or stope in temporarily narrow vein

Stope timbering



Solid timber bulkhead in a square set

Rock-filled crib in a square set

FIGURE 44.—Types of Timber Support for Heavy Ground.

Spiling or forepoling and breast boarding will sustain weak or loose ground, particularly if wet. Pneumatic screw jacks (fig. 50) or scraper hoists with a special device (fig. 51) are used by some companies for driving forepoles or spiling. Several companies in the Lake Superior district use large augers to bore holes at the top of an advancing face in soft ore; each round forepole is placed with one end supported in a hole while the other end rests on the cap of the last timber set. One company drills a series of 2-inch holes spaced 6 to 10 inches apart at the top of an advancing face, then places 1½-inch-diameter lengths of pipe in them with one end supported on the cap of the last set before the round is blasted. When advancing a mine opening in ground requiring support, it is good practice to set overhead forepoles between the last set and face before loading out the cut and setting timbers. Rock bolting has proved to be an effective aid where spiling or forepoling was formerly considered necessary.

Steel H-beams are used by some iron-mining companies to replace steel rail beams formerly used for forepoling. Steel rails have been known to break under sudden shock loads.

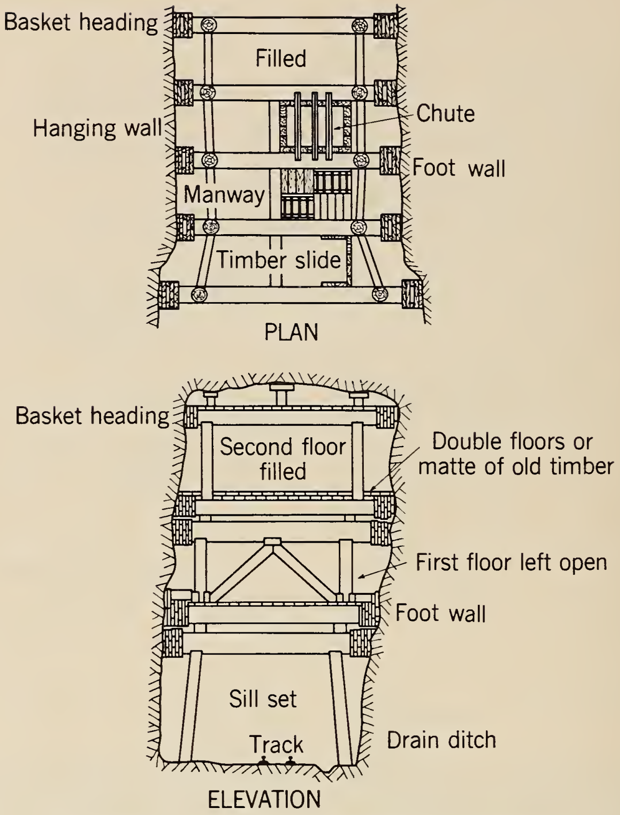


Figure 45.—Method of Placing Stringer Sets.

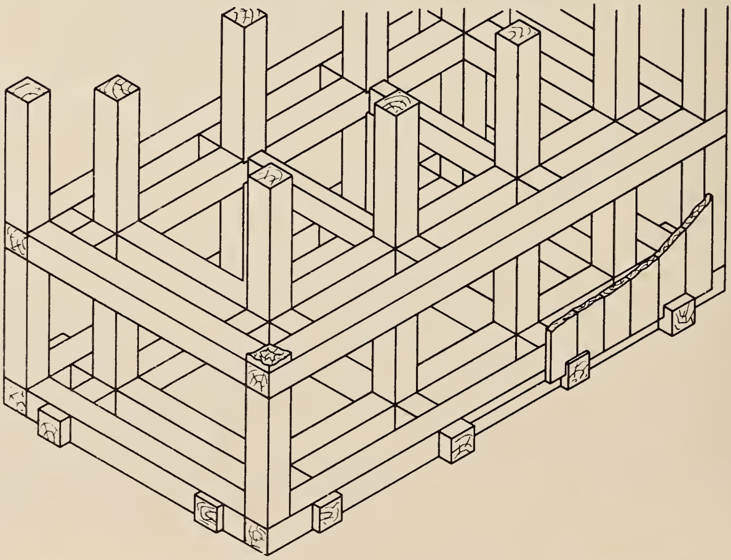


Figure 46.—Shaft Timbering With Squeeze Set.

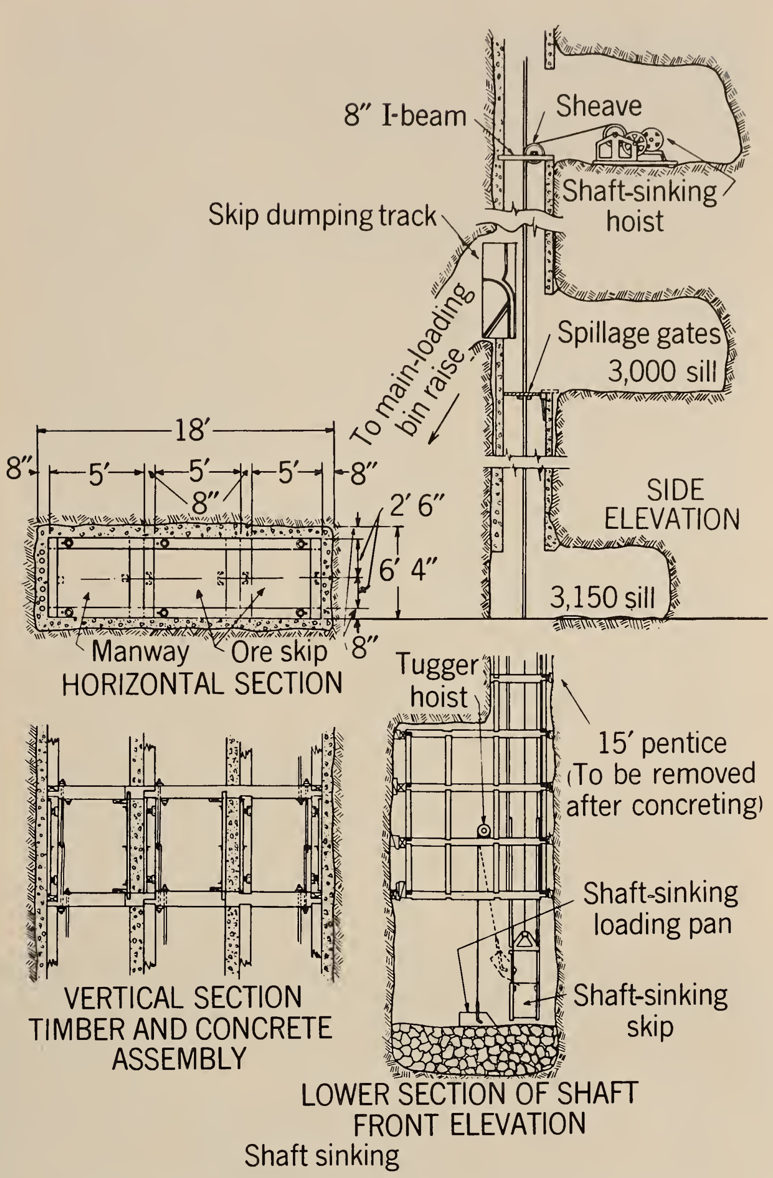
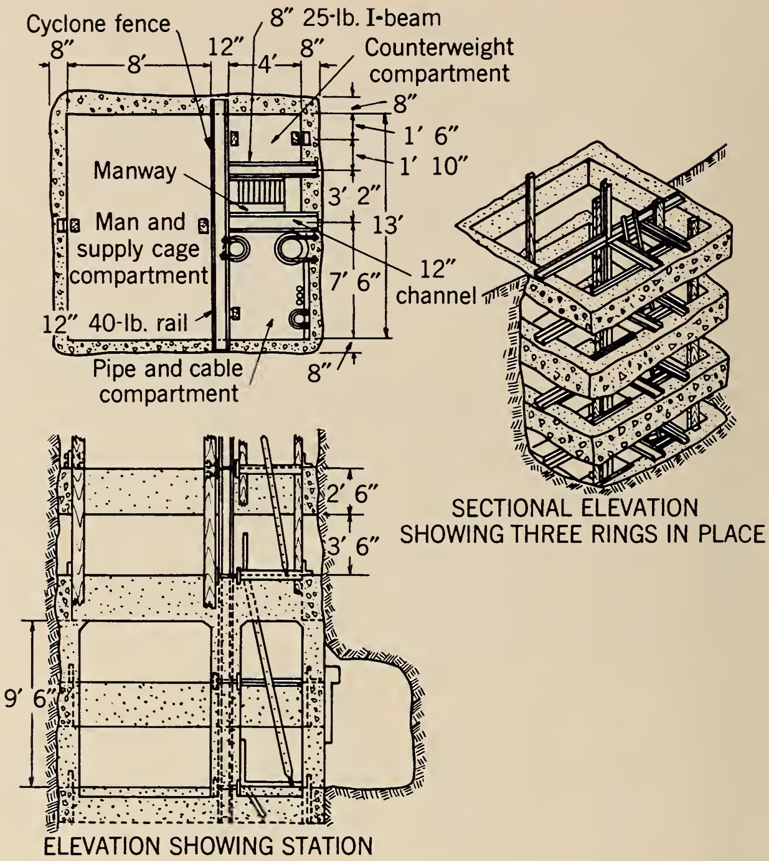


FIGURE 47.—Shaft Sinking; Timber and Concrete Support.



SECTIONAL ELEVATION SHOWING THREE RINGS IN PLACE

ELEVATION SHOWING STATION

Figure 48.—Concrete Rings for Shaft Support.

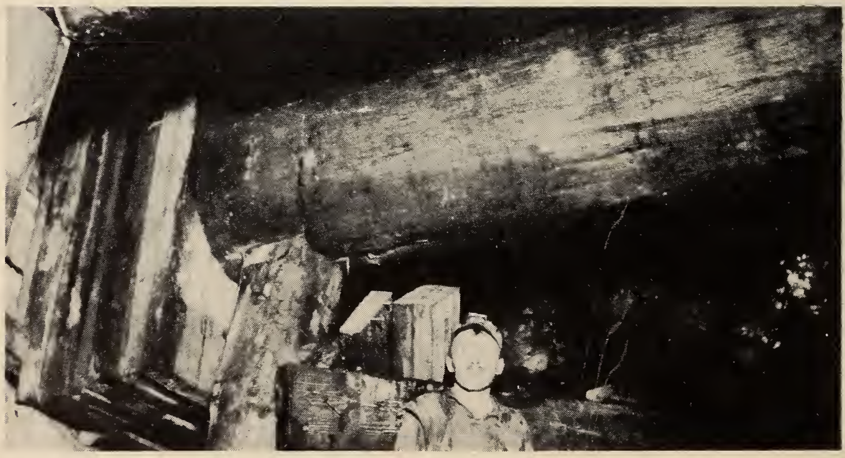


Figure 49.—Heavy Timber and Basket Blocking.

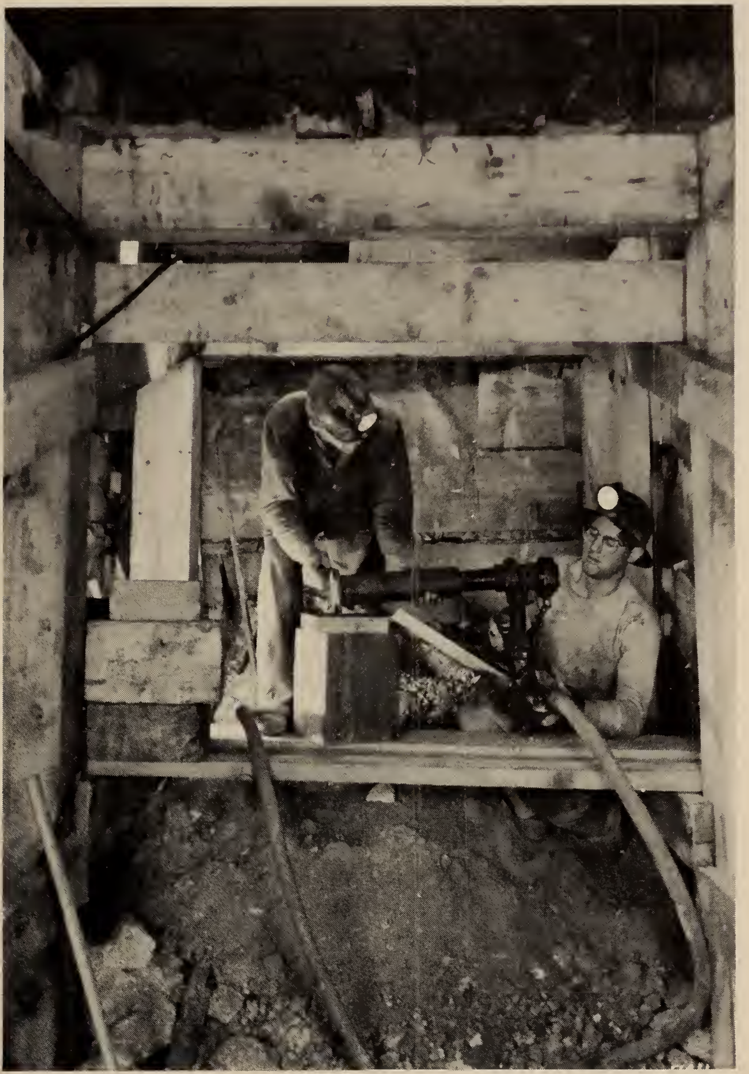


FIGURE 50.—Machine for Driving Spiling.

A precaution to be observed when using breast boards is to have the sets well blocked and braced at top and bottom to prevent having the post kicked out by pressure of the ground. Booms are an excellent means of giving temporary or provisional overhead protection in bad ground until the regular set and lagging can be placed (fig. 52). Care should be taken in selecting the steel for boom stirrups, as the higher carbon steels may snap if subjected to sudden and violent strain.

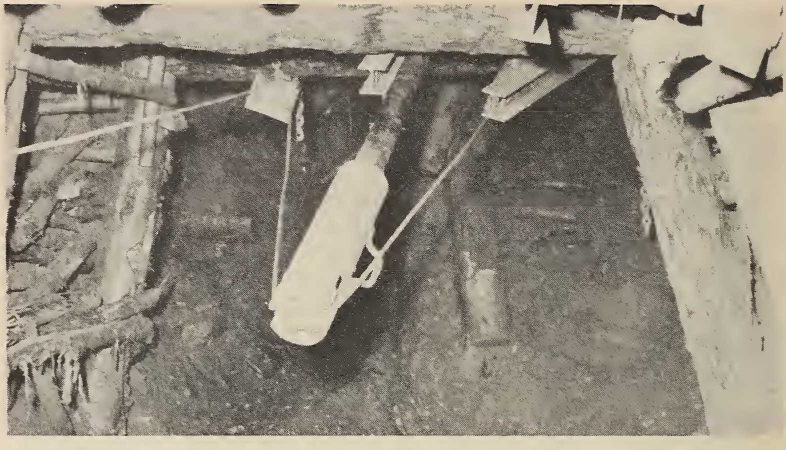


FIGURE 51.—Method of Driving Forepoles and Spiling Using Scraper Hoist.

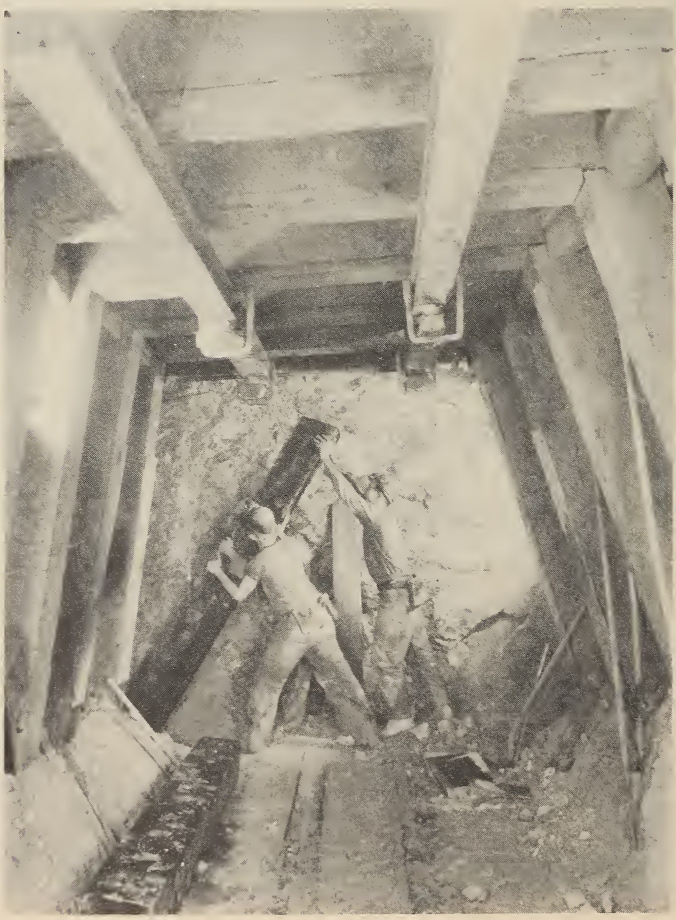


FIGURE 52.—Booms and Lagging Used While Standing Drift Sets.

Cribbing of manways and chutes finds application in filled and shrinkage stopes where heavy pressures are supported by fill or ore and timbers are not blocked to the walls. Cribbing of raises and chutes in narrow veins is customary where the walls of the vein must be supported.

Drift sets for maintaining levels, sublevels, or passageways through stopes may be 3 or 4 piece—4 piece if sills are used. If posts are set vertically, sills become necessary to maintain the posts normal. Some companies have found it advantageous to use a two-piece sill with inclined or battered posts. Drifts carrying a ditch at one side generally use a longer post on that side, and sills are less advantageous in such cases.

In some northeastern United States mines, the backs of development drifts sre supported by lengths of 6-inch steel pipe placed in circumference-conforming saddles, each supported by 2 steel pins placed in holes drilled at proper space and heights in each wall or rib of the drift. Above the caps, round lagging with minimum 4-inch diameter is placed in enough tiers to block the space between the back and the tops of the caps.

Some mines use roof bolts to support steel cap pieces and lagging across backs having long spans in pumprooms, shaft-station cutouts, and other underground rooms.

The chief advantages of square sets are reliability, safety, and flexibility. One of its most notable uses was in the Comstock operations in the early mining days of this country.

Bridge pieces over caps and back of posts and backsetting the lagging offer protection to caps and posts by taking the pressure off the sets.

Where rock will not support itself over the span of the opening, the back may sometimes be arched by blasting it in that form. If timber is used it should be installed before the rock becomes loose and dangerous (figs. 53 and 54).

Cribs and bulkheads are a quick means of building up a support for ground too weak or too heavy to be supported with posts or stulls. Figure 55 shows a rock-filled crib in a deep mine.

The use of concrete and steel support (figs. 56 to 59) is gaining favor; concrete and steel sets and shaft rings, together with metal and prefabricated concrete lagging, eliminate most fire hazards, reduce hazardous repair and maintenance work, do not rot, and offer less resistance to ventilation currents.

Where block caving is practiced at some mines, monolithic concrete (approximately 4,000 p. s. i.) is pressure-pumped between collapsible forms to support the walls, back, and, in some instances, the floors of scram and grizzly drifts. Reinforcing steel is not used because vibration and rock movement tend to be transmitted via such reinforcing to cause failure of the concrete. Many of the newer shafts with finished circular cross sections are supported by high-strength, nonreinforced monolithic concrete placed by using collapsible forms.

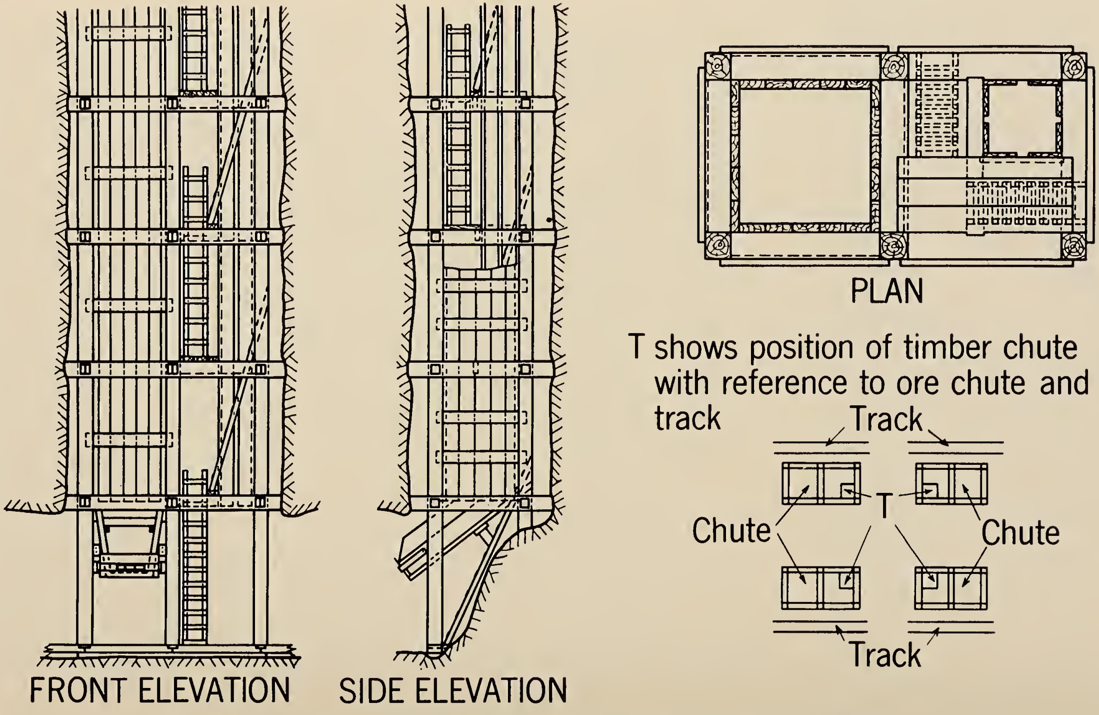

FRONT ELEVATION
SIDE ELEVATION
PLAN
T shows position of timber chute with reference to ore chute and track
Track
Chute
T
Chute
Track

Figure 53.—Standard Six-Post Square-Set Raise With Safety Manway.

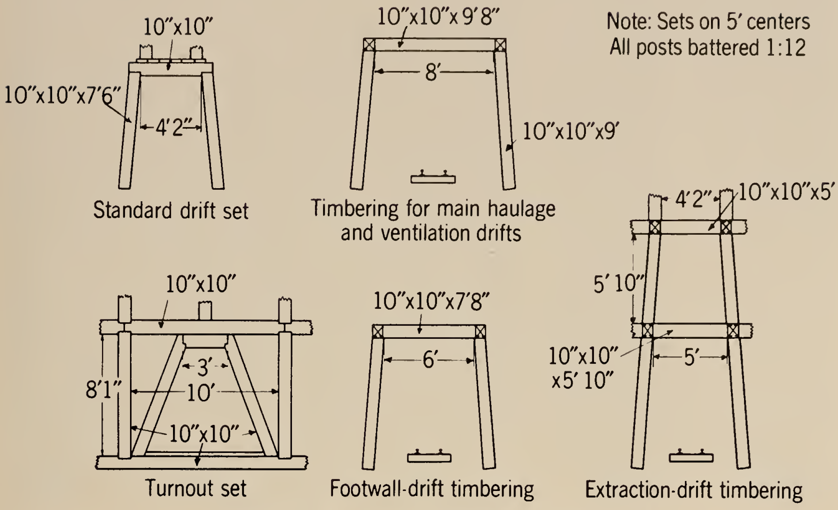


FIGURE 54.—Methods of Timbering.

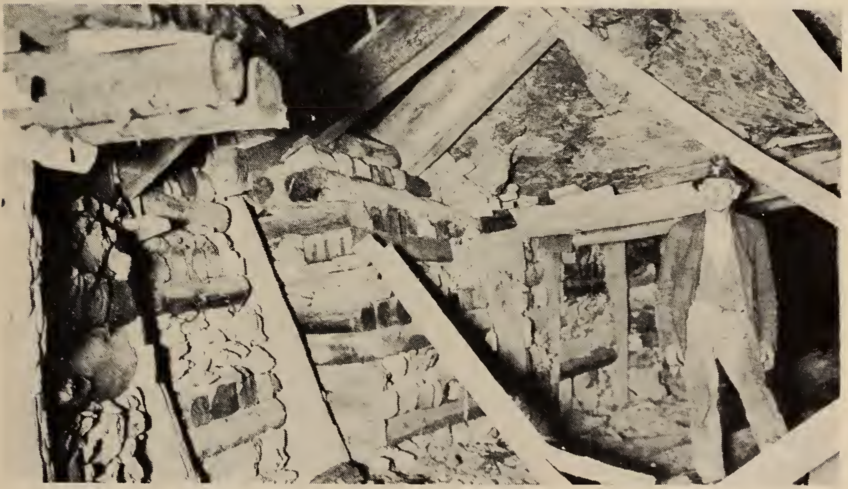


FIGURE 55.—Rock-Filled Crib in Deep Mine.

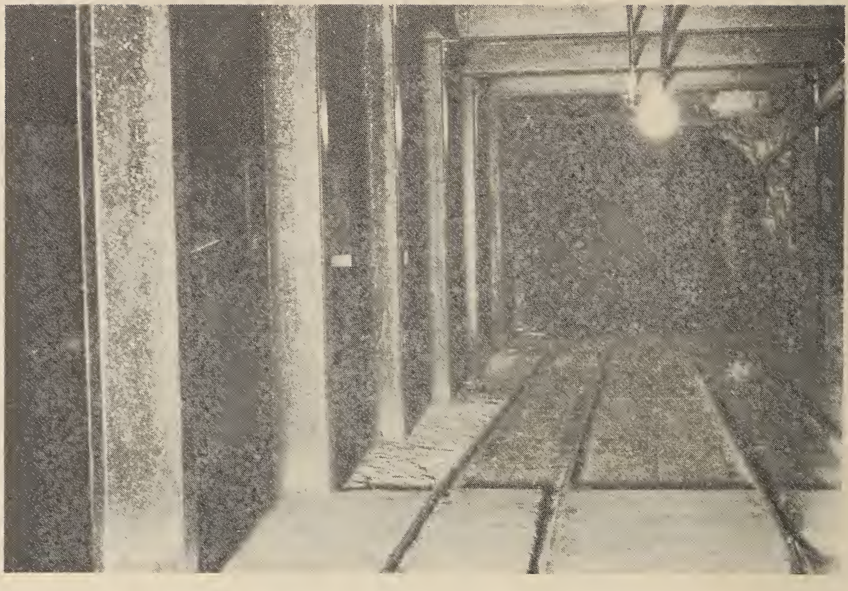


FIGURE 56.—Steel Drift Sets in Main Haulageway.

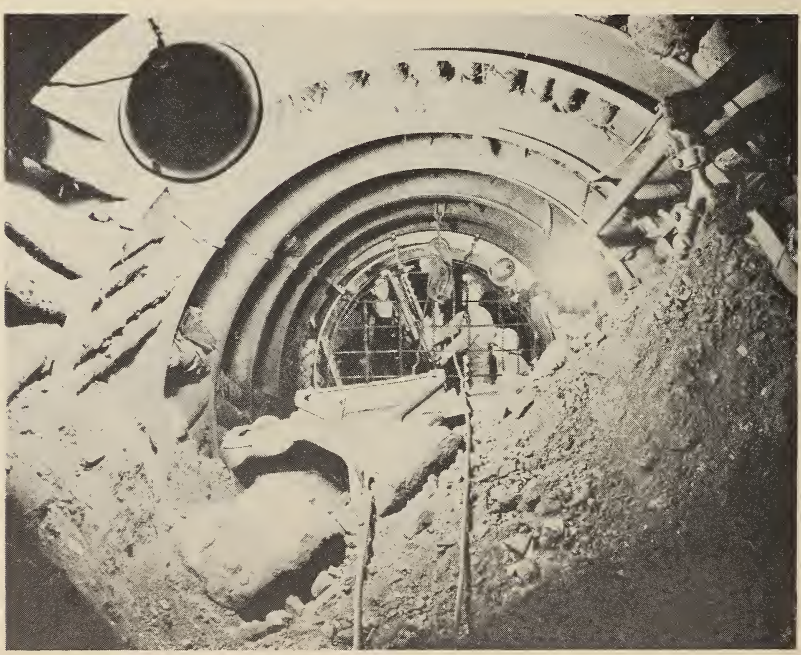


FIGURE 57.—Circular Steel Sets in Scraper Drift.
(Courtesy, The Cleveland-Cliffs Iron Co.)

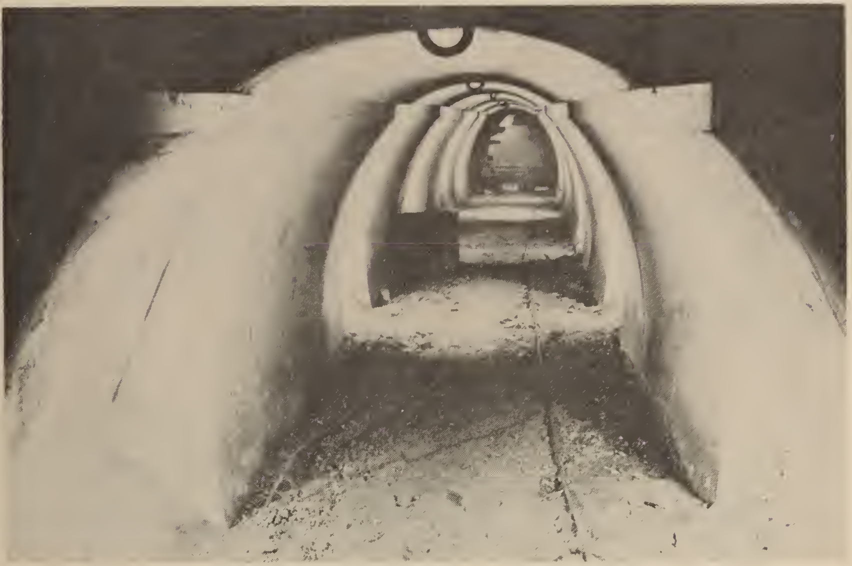


FIGURE 58.—Concrete Supported Scraping Drift.
(Courtesy, The Cleveland-Cliffs Iron Co.)

CAUSES OF LOOSE ROCK

A survey of fatal accident reports shows that a very small number of rock-fall accidents are due to large-scale caving of ground. A large majority are caused by the falls of rocks less than 3 tons in weight. The most common causes given for loosening and falls of these rocks are:

Blasting.—In mines where rock structures are not hard and firm, blasting loosens pieces already separated by slips and fractures.

Faults.—Slips and seams due to faulting may allow release of blocks as other support is removed or weak binding gives away.

Weathering.—"Air slacking" is a common reason given for the presence of loose rock after the walls and back have once been barred and scaled. Air circulation, causing moisture and temperature changes which affect the binding between pieces of rock, can sometimes be held responsible, as can percolating water.

Pressure.—Rock bursts, bumps, and spalling are usually results of weight or pressure of overlying ground.

Area of Excavation.—A common cause of failure and falls of blocks or slab is the opening of too wide areas without enough artificial support.

Time Element.—When pressure is a factor the time during which ground is left open may be an important consideration. When work in stopes or other openings is continuous, there is likely to be relatively little trouble from loose rock, but when operations are slowed or intermittent bad conditions may develop. Where stopes are timbered or rock bolted, this may be less important.

Loose rock may be caused by natural forces or by necessary mining processes. The supervisors of any mine become familiar with the ground in which they work and know which of the preceding factors must be given attention.

ROCK-BURST PROBLEM

A study of rock bursts at one mine in the Lake Superior District [32] showed that most of them occurred shortly after blasting. The only practical solution to the problem at this mine was to increase the interval between working shifts to a minimum of 2 hours with all men out of the mine and to do all blasting by electrical means, with all the blasting lines tied to a common line at the shaft in order to fire simultaneously all the blasts on the various levels and stopes in an effort to trigger rock bursts during the idle period in the mine. This operating procedure has also been found useful in some deep mines in Canada.

The following information on rock bursts has been compiled from reports and from articles in technical publications: [33-37]

Rocks at depth are subjected to a confining pressure, which is due to the weight of the superincumbent load. When the rock will no longer support a specific opening and the rock is competent and brittle, rock bursts take place. The depth at which this condition occurs varies greatly. This is due largely to the physical properties of the rocks and to the size of the underground excavation. The splitting of rocks on the surfaces of mine openings is an early indication that the rocks have been strained beyond their elastic limits. Other factors also must be considered where rock burst conditions are found, such as earth movements and the degree of fracturing within the rocks themselves.

The theoretical background of rock bursts is still in the field of opinion, but there is reason to believe that any mine opening initiates its own stress zone. These stresses reach their maximum around the surface of the opening and the ground immediately surrounding the opening. The stress intensity increases with the size of the excavation, and when two openings approach each other the intensity is increased. Thus, pillars become more susceptible to rock bursts. The stress arrangement is an overlapping of stress zones, with a general stress tendency to merge individual domes into one large dome embracing the whole area.

As compared with gradually reducing ground to the pillar stage by stoping, pillars may be cut out designed for failure, as a pillar, though crushed, retains a useful supporting value. The maximum size of pillar that will crush gradually with stoping must be determined by experience.

Another feature common to pillars before failure is that the stress concentration invariably causes a rearching of development openings passing through them, with the attendant risk of violent bursting. The working of development openings in pillars is dangerous and has been the cause of some very serious accidents.

Shaft pillars impose a special problem. It has been thought, particularly in South Africa, that shafts located in the footwall experience the least difficulty from pressures and rock bursts if the ground overlying them is stoped out first.

It is commonly considered that foot-wall drifts are clear of trouble. Experience has shown that this is not the case and that both foot- and hanging-wall drifts have been seriously damaged by bursting in pillar areas where they come under the influence of overlapping stress zones created by stoping. The same holds true for pumproom chambers and other excavations. A pillar should never be allowed to form in their vicinity unless it is a large pillar designed for their protection. With extensive ore bodies, bursts have occurred in openings as far as 200 feet in the foot wall and 300 feet in the hanging wall.

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[32] Obert, Leonard, Progress Report—Rock-Burst Investigation, Calumet & Hecla Mining Co., Calumet, Mich.: Bureau of Mines Report E 13.1, 1951, 9 pp.

[33] Morrison, R. G. K., Report on the Rock-Burst Situation in Ontario Mines: Rept. to Ontario Mining Assoc., 1940, 43 pp.

[34] Ontario Mining Association, Report of the Rock-Burst Committee: 1941, 15 pp.

[35] Work cited in footnote 29, p. 52.

[36] Spaulding, Jack, Observations on Rock Bursts: Eng. and Min. Jour., vol. 149, May 1948, pp. 91–93.

[37] Johnston, M. L., Pressure and Rock Burst From a Practical Point of View: Eng. and Min. Jour. vol. 151, September 1950, pp. 91–94.

Stoping on two veins toward their junction has frequently resulted in rock bursts. A similar condition develops when parallel veins are stoped simultaneously. The nearer the veins are together, the greater the possibility of intermittent bursting. In the branching veins the solution is to stope the junction first and work away on one leg before commencing regular stoping on the other. In the latter case, 1 parallel vein (preferably the hanging-wall vein) should be stoped well in advance of the other, or the 2 veins should be stoped singly.

In general, control measures to minimize rock bursts center around efforts to establish a gradual growth of doming processes which must go on in deep mining. The sequence of stoping and support must be combined to localize and reduce these effects. When domes form under pressure, all stope supports must be compressible, because to date experience has produced no method of support capable of rigidly withstanding the tremendous rock pressures. Filling following closely behind the stope face is a most effective means of exerting resistance to doming process; however, many well-filled areas have been subjected to severe rock bursts. It is incorrect to assume that rock bursts can be any more than partly controlled even with the best fill.

Figure 59 shows the effect of a small rock burst on steel sets, as compared to timber support, and clearly illustrates the safety advantage of steel support.

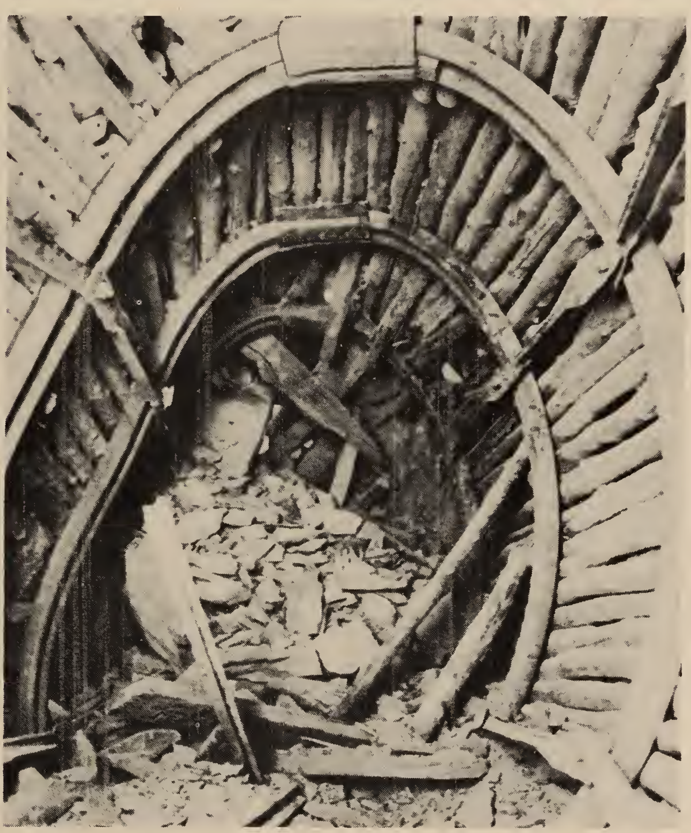

FIGURE 59.—Effect of a Small Rock Burst on Steel Sets as Compared to Timber Ahead.
(Courtesy, R. G. K. Morrison, consulting engineer.)

CAUSES OF ROCK-FALL ACCIDENTS

Common rock-fall accidents are described in the following incidents abstracted from reports of company, State, and Bureau of Mines inspectors and safety personnel:

Two miners were picking down a muck pile in a shrinkage stope; a slab of rock fell from a slip on the hanging wall, rolled down the pile, and crushed one man against the foot wall.

* * *

A miner was drilling a hitch for a post in a timbered drift. One of his partners saw the ground start to move and shouted, but the noise of the machine kept the driller from hearing. About 10 tons fell, killing him instantly.

* * *

A contract miner in an iron mine and his partner were slicing back on a drift; part of the slice had been blasted down but did not fill completely. These men had removed ore blasted from a cut by the night shift, placed timber, and were drilling when the place caved in, crushing and killing one of them.

* * *

A miner had blasted at dinnertime, and his partner (who was trimming the back after the blast) called for a pick. As the miner was bringing the pick, he was struck by ground peeling off the brow.

* * *

Two miners were barring down ore in an open-caving system stope after blasting. Loose ground was noticed, but one miner went under it to get down some good ore, although both had previously and purposely stayed away from that part of the stope. The loose ground fell and crushed the miner's skull.

* * *

A miner on the night shift was drilling in a large 3-compartment raise being driven at an angle of 45° when a large slab fell from the face and killed him. The miner and his partner had previously tested the face and knew a portion was loose and likely to fall when drilled. After installing timber they placed a prop under the loose ground and then proceeded to drill. While the second hole was being drilled, a slab fell and pinned the miner against the post.

* * *

A completed stope dipping at about 35° and from 8 to 30 feet wide had considerable ore left unmined on the footwall. When two miners reentered the stope and began drilling in an attempt to recover this ore, a rock fell from the back 75 feet up the stope, struck the footwall above them, and shattered. Fragments struck and killed one miner. With reasonable care, all the recoverable ore should have been mined during initial stoping, particularly since it was not economically feasible to scale the back after the stope was finished.

* * *

After several attempts to bar down a loose slab in the back of a drift near the face, a miner sounded the slab with his bar, standing directly under it; the loose slab fell, killing the man.

* * *

A truck driver at an open-pit clay mine received fatal injuries when a 120-pound rock fell from the high wall as he stood by his truck while it was being loaded.

* * *

A miner and his partner had finished scaling the back in their stope and were preparing to drill when a rock fell from the footwall, killing the miner. They had failed to scale the footwall side.

* * *

A scraperman standing near the ladderway suffered a fractured leg when he was struck by a chunk falling from the side of an untimbered scraping drift. The area had not been inspected or barred of loose rock for a long period of time.

* * *

A miner was seriously injured while he was drilling near a partly blocked cave opening. He failed to obey the foreman's instructions to prop up the loose ground before drilling.

The accident reports and studies [38] [39] by safety engineers as to the causes of rock-fall accidents disclose causes that are common contributors. They are:

1. Failure to test or inadequate testing of the surfaces at the working place.
2. Faulty supervision due to improper training of bosses or insufficient time to give thorough attention to working places visited.
3. Negligence of experienced miners or mine officials because of previous infrequency of accidents.
4. Improper training or lack of training of miners.
5. Inadequate lighting of working places.
6. Failure to use suitable mining methods or safer up-to-date practices.
7. Negligence in following and enforcing safety rules or lack of safety rules.

PREVENTION OF ROCK-FALL ACCIDENTS

MODIFICATION OF MINING METHOD

Although the type of mining is not the determining factor in the causes of rock-fall accidents, there are instances where the method used is unsafe in the ground being mined. Changes of mining method have been made in a number of mines to find one giving more support to the back or protection to the miner; these may involve a change to a different method or modification of the one in use.

In nearly all mines men must work or pass under overhanging rock. They should not be required or permitted to do so unless it is known that the overhanging rock may be expected to stay in place. On this basis, these accidents are preventable by the proper training of miners and the sincere efforts of mine supervisors.

RESPONSIBILITY OF MANAGEMENT AND SUPERVISORS

After a suitable mining method has been adopted, the management must continue to study the probable or known hazards with a view to eliminating them. Investigations and reports of accidents or dangerous conditions found will be useful if written descriptions and sketches are put before all bosses and occasionally before all the men. Bosses should instruct the men in a standard routine or see that they follow a routine in which instruction has been given. They should examine each working place frequently enough to insure that it is in safe working condition, and if a dangerous condition exists they should stay on the spot until it is corrected. Orders or instructions should be given specifically and not in the form of generalities or of admonitions.

One of the main responsibilities of the mine operating officials in the prevention of accidents from roof falls is to have plenty of suitable timber or other support material readily available to workers to use in trying to make places safe. Mine officials should also experiment with and adopt newer and better methods of roof and wall support where applicable and also provide protective safety equipment for the protection of the underground workman.

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[38] Denny, E. H., Accidents From Falls of Rock or Ore in Metal Mines: Bureau of Mines Miners' Circ. 41, 1940, 21 pp.

[39] American Institute of Mining and Metallurgical Engineers, Health and Safety Committee, Importance of Falling Ground, Rock, and Coal as an Accident Cause: Tech. Pub. 1362, Min. Technol., September 1941, 8 pp.

Routines in use by some well-managed companies include the following steps:

1. After blasting, miners put up the ventilating tube and wet down the face and muck pile from a place protected by timber, rock bolts, or solid rock.
2. They inspect and repair timber, retighten rock bolts, and examine the back, using lights or a system of lighting that will enable them to see clearly. The most dependable type of individual miners' light is the permissible electric cap light.
3. They clear the floor from which the work will start and bar down, remaining under protected or scaled areas while doing so.
4. Where barring down will not make the back or walls safe, stulls or stringers are put in or the loose rock is blasted down before the other work is done.
5. Manways and manway compartments in raises, winzes, and shafts are kept free of loose rock on the walls and on the floors of platforms or sollars.

RESPONSIBILITY OF MINERS

Any man's desire for self-preservation should lead him to stay clear of dangerous places or to remove the danger if he is mindful of its presence and is able to remove it. He may be moved to take a chance, in the belief that time and work may be saved thereby and that his previous immunity will protect him. Unsafe working habits may be continued without accidents for a period sometimes reaching into years, leading to a mistaken idea that the individual is superior to the hazard, thus giving him a false sense of security. However, unsafe working habits ultimately will cause accidents to such a worker, his coworkers, or both. Every miner should be obligated to follow reasonable instructions given as a safeguard against injury or death. Each individual employee owes an obligation to himself, his family, and his employer to use reasonable care in his own behalf. The employer usually pays insurance or compensation for injuries and is entitled to the cooperation of the men in reducing these costs and other penalties of an accident. It is a duty of the miner to help in trying to train himself to find and recognize loose and dangerous ground and to use the methods and safeguards provided to avoid injury from it.

TESTING GROUND AND BARRING DOWN

Sight and sound are used together in detecting loose rock. A pull with the bar often is a good test, but if a piece of rock will not move or come down when pried with a bar it should be examined both by sight and sound; if it can be done safely, vibration tests should be made by putting a hand on the rock and striking the suspected loose rock with a hammer. In testing by sound or vibration, the striking tool must have enough weight to give a dependable amount of vibration. If a suspected loose rock cannot be taken down or proved solid, it should be blasted down, or support should be placed under it without delay. This is important, as scores of fatalities have been due to fall of rock or ore after efforts to pull it down at some previous time had been unsuccessful.

A straight, sharp bar should be used, of a length that will enable a miner to remain out from under the rock on which he is working. If desired, one end may have the form of a pinch bar.

Alloy-metal bars are being used, to give greater length with less weight (figs. 60–63). Short lengths of rubber hose placed above the hands on scaling bars have been effective in reducing hand injuries. Small pieces of rock sliding down the bars are deflected by the hose.

A number of mines and quarries have successfully used specially designed guns (fig. 64) for removing loose rock from mine walls and roof. Pieces of rock ranging up to 3 tons have been shot down from heights up to 120 feet by 3-ounce soft-lead projectiles. Other innovations for scaling are telescopic lift scalers (fig. 65) and traveling cranes with self-leveling platforms (fig. 66).

Testing should be done from time to time during the shift, as well as on starting work after blasting; this is especially true as to overhead rock surfaces that have been relatively recently exposed to air.

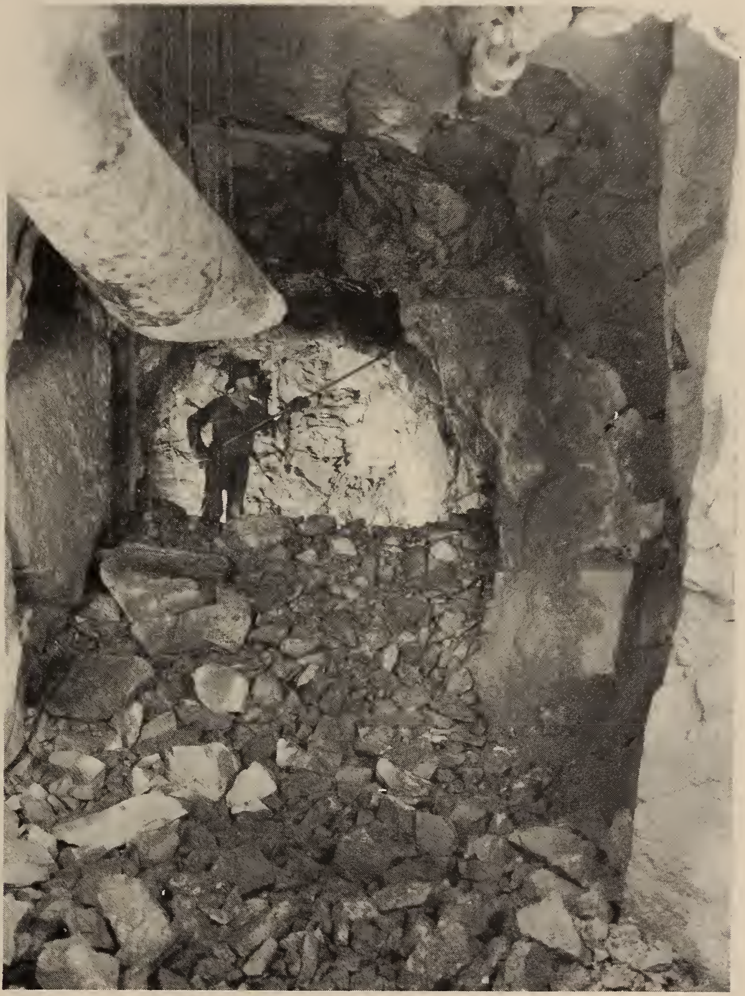

FIGURE 60.—Barring Down in Drift Face.

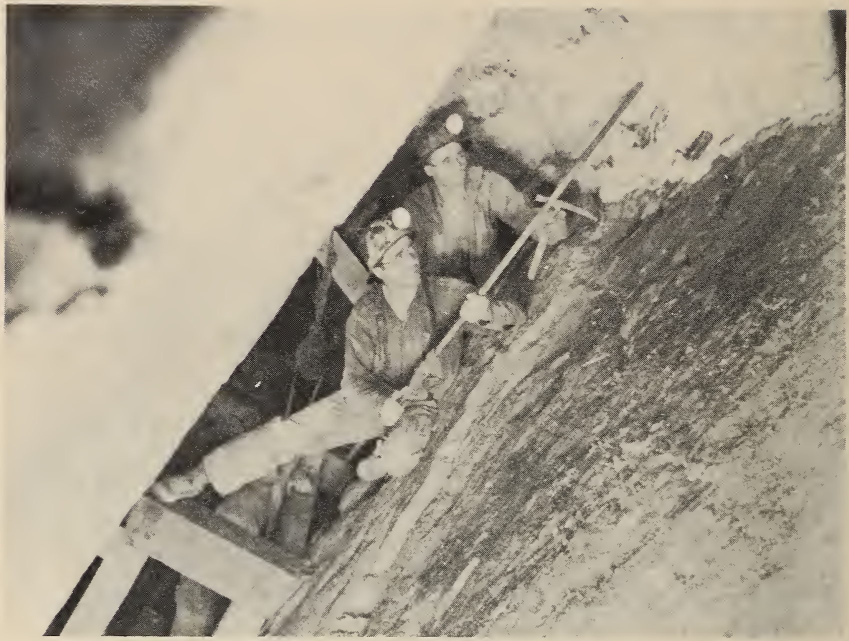

FIGURE 61.—Barring Down Face of Raise in Narrow Vein.

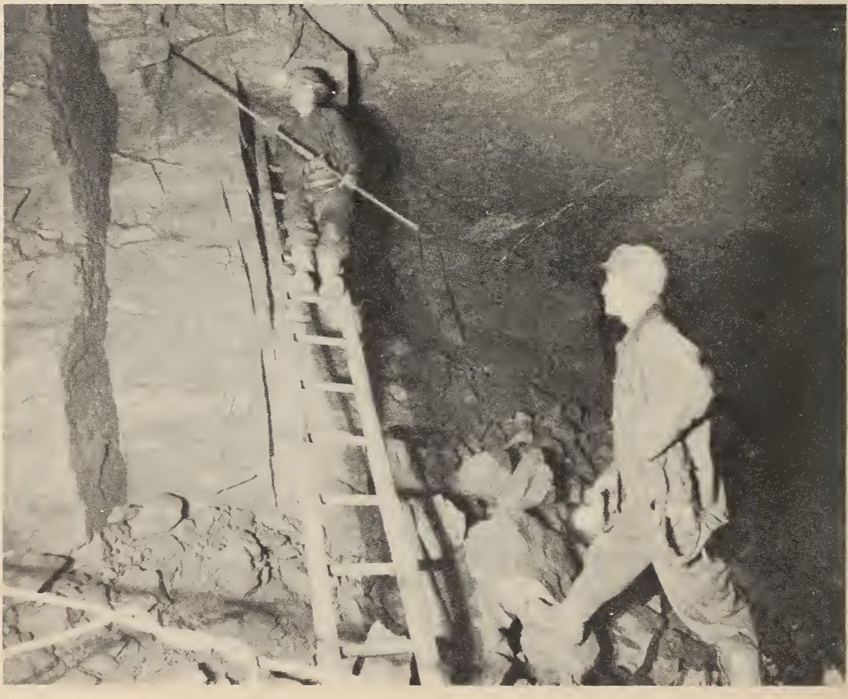

FIGURE 62.—Barring Down Face of Stope in Hard Ore.

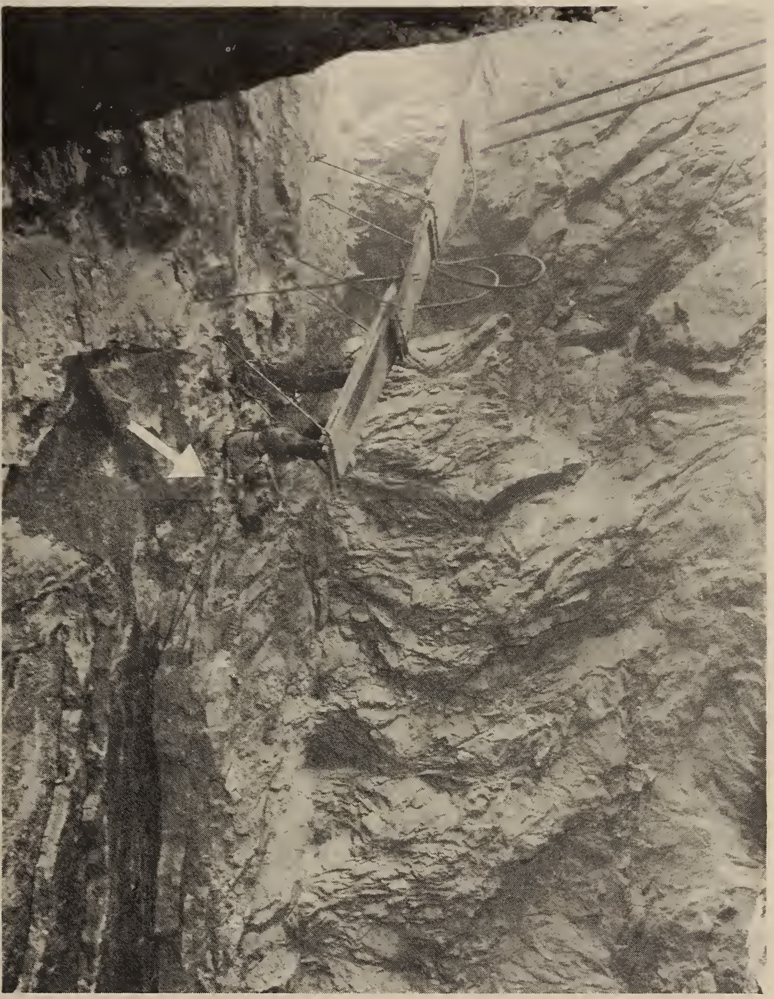

FIGURE 63.—Arrow Points to Men Barring Down From a Trapeze in a High Open Stope.

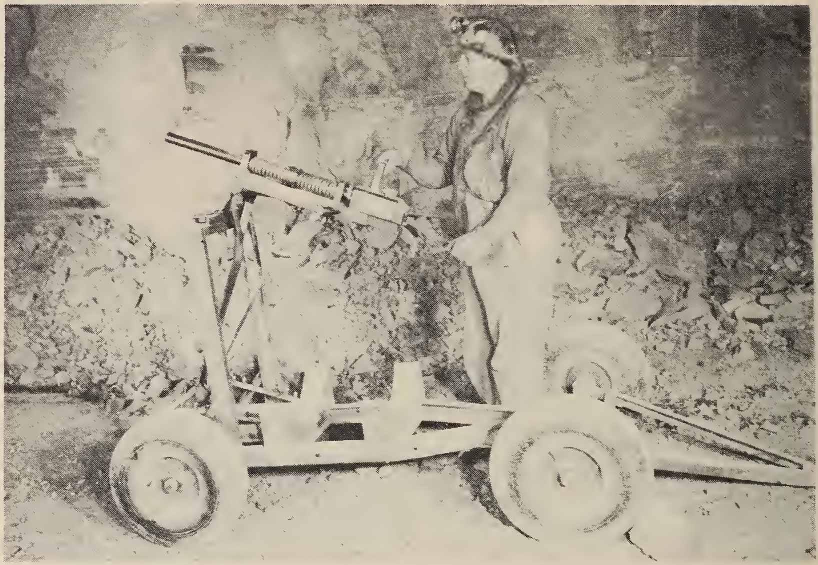

Figure 64.—Industrial Gun Used for Removing Loose Rocks on Mine Walls.

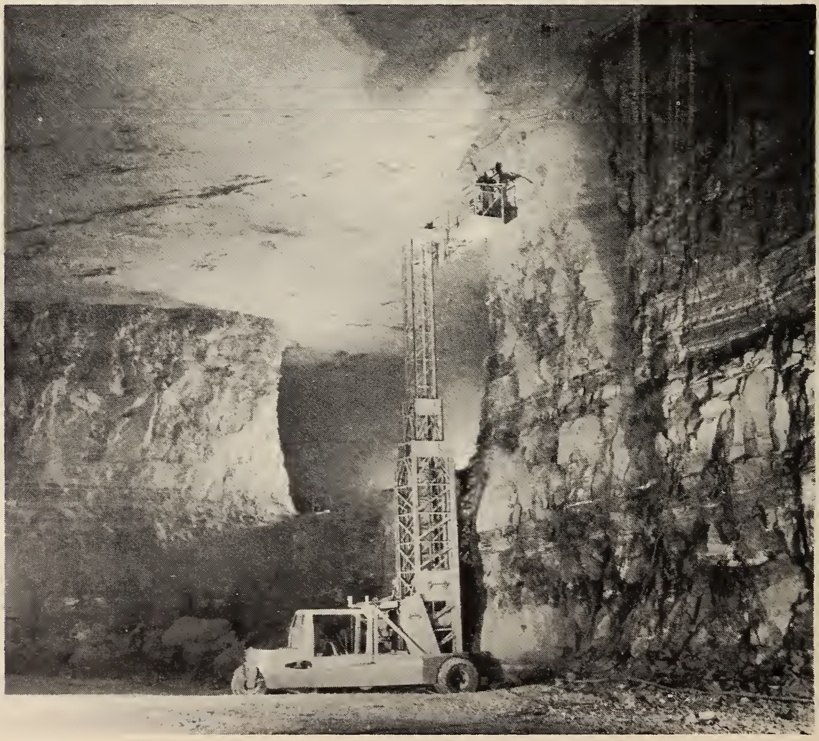

Figure 65.—Using Telescopic Lift Scaler in Rooms 50 Feet High.

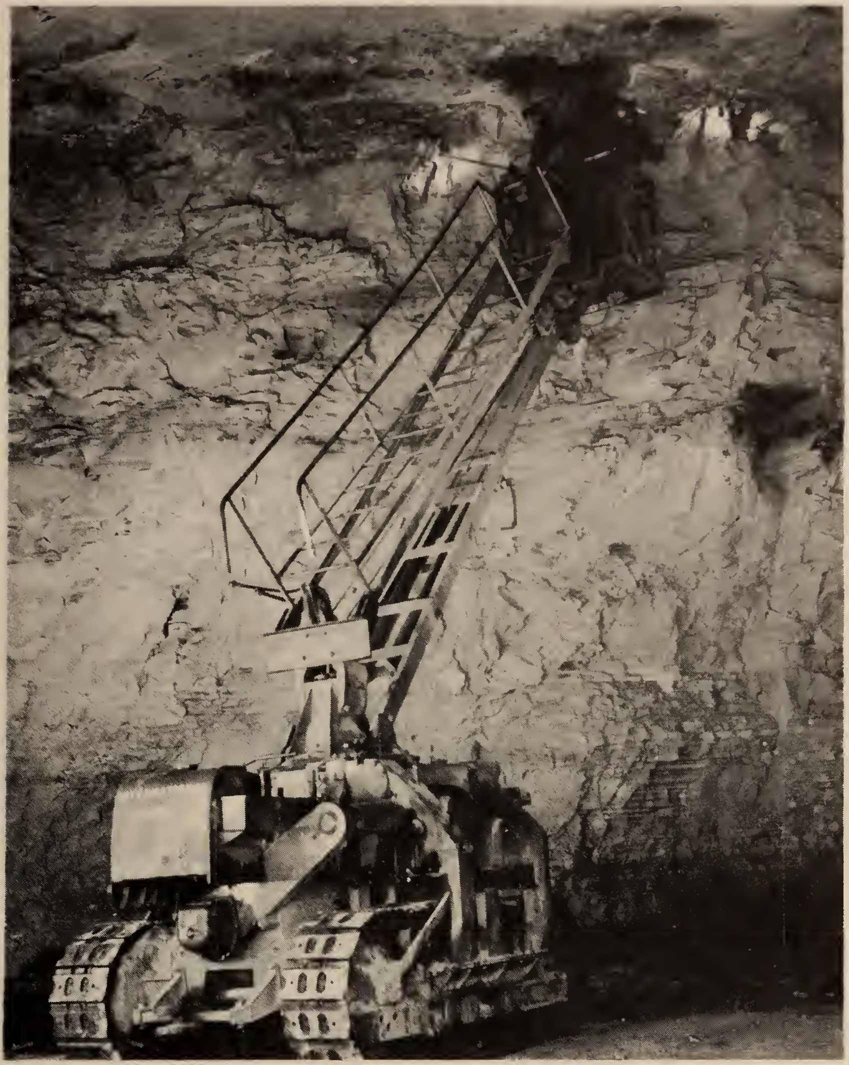

Figure 66.—Traveling Crane With Self-Leveling Platform.

PROTECTIVE EQUIPMENT

The use of protective equipment, such as hard hats, goggles, gloves, and safety-toe shoes or boots, has been effective against injuries from rock falls. The use of hard hats has become customary by persons who are well-informed on accident occurrence, and hundreds of lives have been saved by them in addition to lesser injuries that have been prevented by their use. Safety-toe shoes and boots have helped to reduce the smaller injuries from falling pieces of rock, and goggles have prevented hundreds of persons from losing their eyesight, through warding off relatively small, high-velocity pieces of rock or ore. Many mining companies require stope miners who work on benches in open stopes to wear safety belts with ropes attached to eyepins fastened in solid ground. In event of sloughing of the benches, missteps, or other causes where the men might fall into the stope, the safety belts and ropes are designed to limit their fall.

To protect men from falls of rock while climbing raises, a protective metal deflecting shield worn on the back (fig. 67) has been introduced and used successfully. A hook and safety belt is an added feature to protect miners from being knocked off the ladder by severe blows.

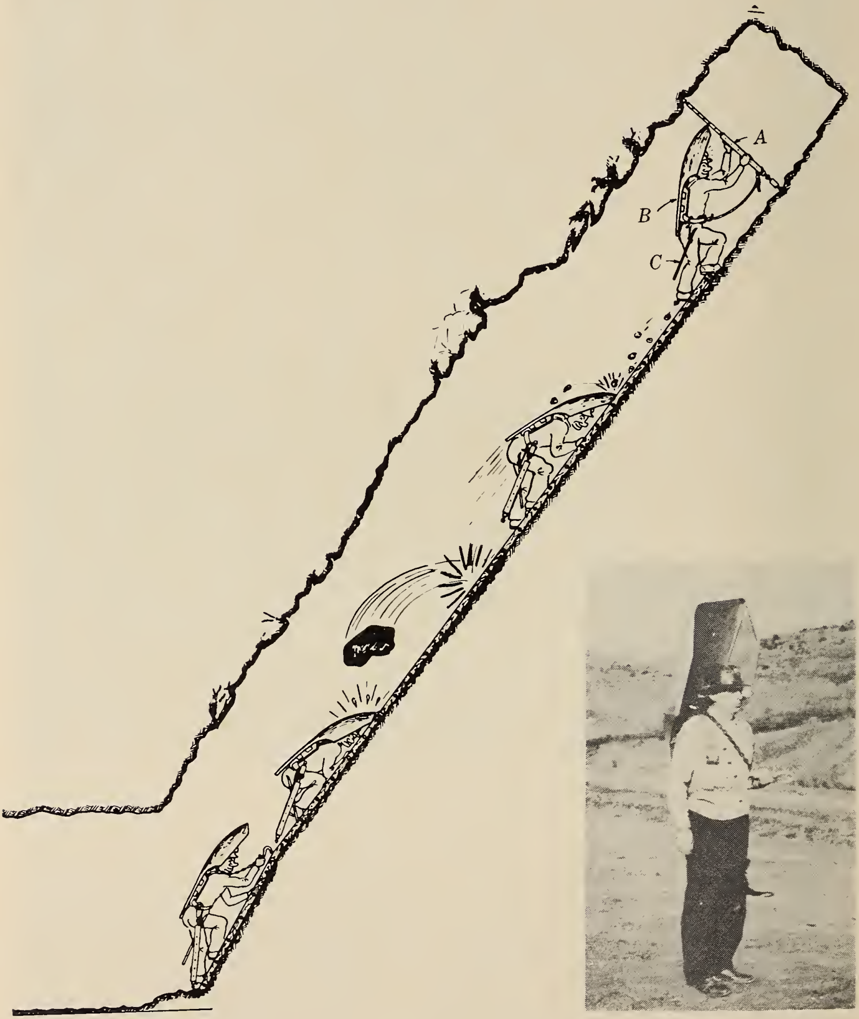

Figure 67.—"Beetleback" Shield for Protection From Falling Rock While Climbing Raises. *A*, Jack bar; *B*, beetleback; *C*, steel bar. (Courtesy, Lamar Weaver and Tennessee Copper Co.)

TRAINING MINERS AND BOSSES

Any mining organization that sets out to reduce rock-fall accidents is very likely to succeed in its efforts by determining the causes of these accidents, making needed changes in methods where it is possible to do so, and carrying out systematic education of bosses and miners to insure observance of well-thought-out safety practices that are adopted. Statistics show that those companies that have a planned educational and safety program, with regularly held meetings involving all personnel, have the best safety records.

Cooperation of management, bosses, and men has repeatedly been obtained by making all of them acquainted with the purposes of the program and the means to be used in carrying it out. This may be done by meetings of various groups where matters are explained and discussed, by personal contact, by instructions on the job, and by studying available literature. At some mines stope schools are held for new employees, with a followup after they start regular work. At other mines, talks and illustrated lectures are held on company time. Modern "job-training" techniques are productive of safe and efficient mining. Information circulars on accidents or other current subjects are issued by safety departments at various mines, as well as by some State agencies and by the Federal Bureau of Mines.

CONCLUSIONS

Rock-fall accidents cause more fatalities in mines than any other single factor. The prevention of accidents of this type requires ceaseless vigilance and constant effort on the part of both supervisors and workmen in cooperation with inspection agencies and management. It is management's responsibility to provide miners with as safe working conditions as possible; with adequate training, tools, and materials; and with competent supervision. The impetus for safety consciousness rests with the mine management, the safety department, the mine supervisors, and, to a limited extent, the State or other Government inspection force. The mining industry and regulatory agencies have done much to formulate safety rules and regulations to reduce rock-fall accidents; however, the main portion of responsibility for this type of mine accident rests with every underground worker once he has learned his trade, because it is impossible for a mine boss to be continually at his side trying to safeguard him.

APPENDIX

SUGGESTED TOPICS FOR CLASS DISCUSSION

1. What are possible reasons for the reduction in accidents from rock falls in the last 20 years?
2. What effect does the choice of a mining method have on the danger from falls of ground?
3. In what kinds of ground are open stopes suitable? shrinkage stopes? cut-and-fill stopes?
4. What are the underlying causes of accidents from loose rock?
5. What responsibility does the boss have if he finds men working under bad ground or hazardous conditions?
6. In what phases of mining operations do most of the accidents caused by fall of ground occur?
7. What are the best ways to overcome this?
8. To what extent are mine officials responsible for accidents caused by falls of ground?
9. What disciplinary measures should be adopted for a careless miner?
10. In leasing operations, who is responsible for maintaining safe working conditions?
11. How should posts or stulls be set with reference to the inclination or dip of the working place?
12. Do headboards and footboards increase the supporting value of posts and stulls?
13. How do steel and concrete compare with timber for the support of ground?
14. How do rock bolts support ground?
15. What are some of the safety and economic benefits to be expected from rock bolting?
16. Should those who install rock bolts be trained? Why?
17. When is the best time to install rock bolts? Why?
18. What are the proper methods for testing the back or face for loose ground?
19. In your experience are more men injured by loose rocks that had been detected or by those not detected or inspected?
20. What precautions are necessary in barring down?
21. Is a rock safe if a few trys will not pull it down?

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18. **Obert, Leonard.** Progress Report—Rock-Burst Investigation, Calumet & Hecla Mining Co., Calumet, Mich. Bureau of Mines Rept. E 13.1, 1951, 9 pp.
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NOTE.—Circulars marked with a dagger (†) are out of print but may be consulted in many libraries. Others may be obtained from the Superintendent of Documents, Government Printing Office, Washington 25, D. C., at the price indicated. Stamps are not accepted in payment for such purchases.

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